This is a digital copy of a book that was preserved for generations on library shelves before it was carefully scanned by Google as part of a project to make the world's books discoverable online.

It has survived long enough for the copyright to expire and the book to enter the public domain. A public domain book is one that was never subject to copyright or whose legal copyright term has expired. Whether a book is in the public domain may vary country to country. Public domain books are our gateways to the past, representing a wealth of history, culture and knowledge that's often difficult to discover.

Marks, notations and other marginalia present in the original volume will appear in this file - a reminder of this book's long journey from the publisher to a library and finally to you.

Usage guidelines

Google is proud to partner with libraries to digitize public domain materials and make them widely accessible. Public domain books belong to the public and we are merely their custodians. Nevertheless, this work is expensive, so in order to keep providing this resource, we have taken steps to prevent abuse by commercial parties, including placing technical restrictions on automated querying.

We also ask that you:

+ *Make non-commercial use of the files* We designed Google Book Search for use by individuals, and we request that you use these files for personal, non-commercial purposes.

+ *Refrain from automated querying* Do not send automated queries of any sort to Google's system: If you are conducting research on machine translation, optical character recognition or other areas where access to a large amount of text is helpful, please contact us. We encourage the use of public domain materials for these purposes and may be able to help.

+ *Maintain attribution* The Google "watermark" you see on each file is essential for informing people about this project and helping them find additional materials through Google Book Search. Please do not remove it.

+ *Keep it legal* Whatever your use, remember that you are responsible for ensuring that what you are doing is legal. Do not assume that just because we believe a book is in the public domain for users in the United States, that the work is also in the public domain for users in other countries. Whether a book is still in copyright varies from country to country, and we can't offer guidance on whether any specific use of any specific book is allowed. Please do not assume that a book's appearance in Google Book Search means it can be used in any manner anywhere in the world. Copyright infringement liability can be quite severe.

About Google Book Search

Google's mission is to organize the world's information and to make it universally accessible and useful. Google Book Search helps readers discover the world's books while helping authors and publishers reach new audiences. You can search through the full text of this book on the web at `http://books.google.com/`

University of Wisconsin
LIBRARY
Class MKD
Book ST4

17582

UNIVERSITY OF WISCONSIN,
DEPARTMENT OF
METALLURGY AND ASSAYING.
No. 123
Rec'd May 8, 1889.

THE

LIXIVIATION OF SILVER-ORES

WITH

HYPOSULPHITE SOLUTIONS,

WITH

SPECIAL REFERENCE TO THE RUSSELL PROCESS.

BY

CARL A. STETEFELDT.

THE SCIENTIFIC PUBLISHING COMPANY,
PUBLISHERS.
27 Park Place, NEW YORK. P. O. Box 1833.
1888.

Copyright, 1888.
C. A. STETEFELDT.

Press of Tuttle, Morehouse & Taylor, New Haven, Conn.

17582

1545584

MKK
·S74

ERRATA.

The Lixiviation of Silver-Ores with Hyposulphite Solutions.

A mistake has been made in the figures of § 322, which should be corrected as follows:

On page ~~46~~ 45 read: "For the production of 100 pounds of this salt, 242.9 pounds sodium hyposulphite and 133.3 pounds copper sulphate are required; a total of 376.2 pounds chemicals."

Lower down substitute: 376.2 pounds for 279 pounds; 201 pounds for 160 pounds; and 16974 pounds for 20900 pounds.

On page 46 read: "0.75 pounds of the $\frac{2}{3}$ salt of Lenz are equal in effect to 1 pound copper sulphate."

And lower down: "while 0.75 pounds of the salt of Lenz add 0.83 pounds sodium hyposulphite to the stock-solution."

Press of Tuttle, Morehouse & Taylor, New Haven, Conn.

17582

1545584

MKK
·S54

To his
Teacher and Friend,
PROFESSOR BRUNO KERL,
of Berlin,
This Treatise is Dedicated
by
His Grateful Pupil,
THE AUTHOR.

PREFACE.

This treatise has been written with the endeavor to fill a gap in metallurgical literature, and to present to the profession a clear, complete description of the lixiviation-process in its most improved, modern form. The older European standard-works on metallurgy give the obsolete practice of lixiviation with hyposulphite solutions, as executed by von Patera and Kiss, and do not enter at all into the chemistry of the process. More recent publications are open to the same censure, and all valuable literature on the subject is scattered through the Transactions of the American Institute of Mining Engineers, the Engineering and Mining Journal, the Mining and Scientific Press, and other periodicals. Improvements in the process have been so rapid, that only an entirely new presentation of facts and observations can do justice to the subject. In writing this book, I have freely used my own papers on the Russell process, published in the Trans. of the Am. Inst. of M. E., Volumes XIII. and XV., but augmented and cast into new form most of the chapters. Mr. Ellsworth Daggett's paper, which will appear in Vol. XVI. of the Transactions, has placed all of Mr. Russell's recent experience and observations at my disposal. My own researches were carried out in the laboratory of Dr. Gideon E. Moore, to whom I am under great obligations. I have investigated myself all the important reactions involved in the lixiviation-process, correcting many inaccuracies and errors which occur in my first paper on the Russell process. This paper had been written from Mr. Russell's notes, and not from personal observations. Notes on the solubility of various salts are taken from Dr. Rudolf Biedermann's Chemiker Kalender, when no other authority is given. Special prominence has been accorded to the Russell process, since lixiviation of to-day is essentially the Russell process in one or in all its features, distinguishing it from ordinary lixiviation.

I found it a most difficult task to obtain correct statistics of the lixiviation-process, and had to confine myself to the reproduction of those contained in Mr. Daggett's paper on the Russell process. Many persons consider it too onerous to keep correct statistics; others do not seem to have a clear idea how to perform this work; while some are adverse to have their results published. I hoped to get statistics from two successful lixiviation-mills, using the ordinary process, namely, the Mount Cory and the Cortez, but my request from the former did not meet with a response, while from the latter the data were so incomplete that their publication would be of no interest whatever. The results recorded in Mr. Daggett's paper should not be judged by themselves, or as fair examples of what lixiviation is and can perform under more favorable circumstances, both as to perfection of plant and character of ore. They should be viewed with consideration of all the local difficulties, and compared with the results of amalgamation which were or could be obtained under the same conditions. Their importance and interest lies principally in this, that they show the great superiority of the Russell process as compared with ordinary lixiviation. I expect, before long, to issue supplements to this volume that will place the statistics of the lixiviation-process on the same sound basis upon which its chemistry rests.

In regard to the separation of lead from hyposulphite solutions by caustic lime, which process was in use at the Mount Cory Mill, Nevada, I had the promise of obtaining a full account; even this remained but a promise. Indeed, in writing this book, I need not extend thanks to the profession for assistance or information privately conveyed.

I have divided this treatise into two parts, preceded by an introduction. The first part treats of the chemistry of lixiviation; the second part of the practical execution of the process. Part I. is divided into 7, and Part II. into 8 chapters. These chapters, and their paragraphs, are numbered in a manner suggested by Dr. R. W. Raymond. In place of numbering the first chapter of the second part I. or VIII., it is numbered XI. The paragraphs in the introduction commence with § 1. Reaching the first chapter of Part I., the first paragraph of this chapter is numbered § 101; the second chapter commences with § 201; the third with § 301, and so on. The first chapter of Part II., that is Chapter XI.,

commences with § 1101. The convenience for reference is at once evident. If I quote §1406, it implies that the subject referred to is in Part II., Chapter XIV., which is the fourth chapter of this part, and in the sixth paragraph of the chapter. In the table of contents the page for each paragraph is easily found. In the same way I have given to all tables the numbers of the paragraphs to which they belong.

In dividing the chapters into paragraphs, it became necessary, for the sake of reference, to make a greater number of subdivisions than a strictly scientific classification would warrant. To remedy this, I use also subdivisions as they should be properly made. The two systems do not interfere with each other in the slightest degree, even when they are, occasionally, merged into one.

Throughout this book I have attached to certain terms, used promiscuously by other writers, a definite meaning. If I speak of "leaching," this always refers to the extraction of soluble salts from roasted ores by water. "Lixiviation" refers only to treatment with hyposulphite solutions. The term "vat," is exclusively used for the vessel in which ore is leached and lixiviated; hence, it is both a leaching-vat and a lixiviation-vat, but generally designated as the latter. "Tank" always means a vessel in which solutions are stored, mixed, or prepared. A "sump" is a vessel for collecting solutions which are pumped to a higher level. All oblong, rectangular wooden boxes, for conveying wash-water, solutions, or sulphides, are called "troughs." The square wooden frames with perforated tops or bottoms, which either act as strainers, or break the force of a current of solution, or serve as receptacles for chemicals to be dissolved, are called "boxes;" in the latter case they are "chemical boxes." Other terms will be explained in the text of the book.

For $Na_2S_2O_3+5aq.$, I have retained the old term of sodium hyposulphite, this salt being commercially known as hyposulphite of soda.

THE AUTHOR.

NEW YORK, July, 1888.
18 Broadway.

CONTENTS.

INTRODUCTION.

Page.

§ 1. History of the Lixiviation-Process 1
§ 2. Defects of the Old Lixiviation-Process 2
§ 3. The Russell Process 3
§ 4. Differences between the Russell Process and Ordinary Lixiviation 4
§ 5. The Russell Process Compared with Amalgamation 4
§ 6. History of the Russell Process 5
§ 7. Ores Suitable to Treatment by Lixiviation 8
§ 8. The Operations Preceding Lixiviation 8
§ 9. The Effect of Wetting Down Roasted Ore while Red-Hot 10
§ 10. The Stetefeldt Furnace 10

PART I.

The Chemistry of the Lixiviation-Process.

CHAPTER I.

The Chemicals.

§ 101. Sodium Hyposulphite 13
§ 102. Calcium Hyposulphite 14
§ 103. Sodium Carbonate 14
§ 104. Caustic Soda 15
§ 105. Commercial Rating of Soda and Caustic Soda 15
§ 106. Sulphur 16
§ 107. Sulphuric Acid 16
§ 108. Prices of Chemicals 16

THE CHEMICAL REACTIONS.

CHAPTER II.

THE REACTIONS OF SODIUM HYPOSULPHITE SOLUTIONS.

Page.
§ 201. Argentic Hyposulphite Double-Salts 18
§ 202. Double-Salts of Lead and Sodium Hyposulphite 18

SOLUBILITIES OF METALS AND VARIOUS COMPOUNDS IN SODIUM HYPOSULPHITE SOLUTIONS.

§ 203. Solubility of Metallic Silver 19
§ 204. Solubility of Metallic Gold 19
§ 205. The Solubility of Silver and Gold in Sodium Hyposulphite Solutions Explained 20
§ 206. Solubility of Metallic Copper 20
§ 207. Solubility of Metallic Lead 21
§ 208. Solubility of Silver Oxide 21
§ 209. Solubility of Silver Sulphide 21
§ 210. Solubility of Gold Sulphide 21
§ 211. Solubility of Silver Arsenate 22
§ 212. Solubility of Silver Antimonate 22
§ 213. Practical Importance of the Solubility of Silver Arsenate and Antimonate in Sodium Hyposulphite Solutions 23
§ 214. Solubility of Silver Chloride 23
§ 215. Influence of Lead Sulphate on the Solubility of Silver Chloride 24
§ 216. Influence of Calcium Sulphate on the Solubility of Silver Chloride 25
§ 217. Influence of Sodium Sulphate on the Solubility of Silver Chloride 25
§ 218. The Effect of Caustic Alkalies and Alkaline Earths upon the Solubility of Silver Compounds 25
Neutralizing a caustic Stock-Solution with Sulphuric Acid.
§ 219. Neutralizing a Caustic Stock-Solution with Sodium Bi-Carbonate 27
§ 220. The Effect of a Caustic Stock-Solution Explained 27
§ 221. Solubility of the Carbonates of Silver, Copper, Lead, and other Metals 28
The Separation of Lead from Silver and Copper by Precipitation with Sodium Carbonate.
§ 222. Solubility of Lead Hydroxide and Cuprous Hydroxide 30
Separation of Lead from Silver and Copper by Caustic Lime.

Page.
§ 223. Solubility of the Oxides and Hydroxides of Copper, Iron, Zinc, and Manganese 31
§ 224. Solubility of Cuprous Chloride 31
§ 225. Solubility of Lead Sulphate 31
§ 226. Solubility of Lead Antimonate 32
§ 227. Solubility of Calcium Sulphate 32
§ 228. Solubility of Caustic Lime 33
§ 229. Solubility of Sodium Antimonate 34

CHAPTER III.

The Reactions of Extra-Solutions.

§ 301. Cuprous Hyposulphite Double-Salts 35
§ 302. The Standard Extra-Solution 35

I. Reactions of Extra-Solutions with Caustic Soda and Sodium Carbonate.

A. Extra-Solution of Standard Composition, containing for 1 part $CuSO_4+5aq.$, $2\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$

§ 303. Reactions with Caustic Soda 36
§ 304. Reactions with Sodium Carbonate 37

B. The Extra-Solution contains for 1 part $CuSO_4+5aq.$, $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$

§ 305. Reactions with Caustic Soda 37
§ 306. Reactions with Sodium Carbonate 38

II. Decomposition of Extra-Solution by Heating.

§ 307. A. Extra-Solutions of Standard Composition 38
§ 308. B. Extra-Solutions containing for 1 part $CuSO_4+5aq.$, $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$ 38

III. Decomposition at Ordinary Temperature.

A. Extra-Solution of Standard Composition.

§ 309. Solutions without free Acid 39
§ 310. Acid Solutions 39
§ 311. Solutions to which Sodium Carbonate has been added 40

B. Extra-Solutions containing for 1 part $CuSO_4+5aq.$, $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$

§ 312. Solutions without free Acid 40
§ 313. Acid Solutions 40

Page
§ 314. Solutions to which Sodium Carbonate has been added ... 41
§ 315. Solutions to which Caustic Soda has been added 41
§ 316. The Difference in Reactions Explained 41
§ 317. Conclusions of Practical Importance 42
§ 318. Investigation on the Formation of Sulphates in the Decomposition of Extra-Solution 43
§ 319. Silver Tetrathionate 43
§ 320. The Effect of Extra-Solutions upon Silver-Compounds and Silver-Minerals, compared with Amalgamation 44
§ 321. Extra-Solution with Cuprous Chloride 44
§ 322. Extra-Solution made by dissolving the $\frac{2}{3}$ salt of Lenz in Sodium Hyposulphite 45

SOLUBILITIES OF SILVER, GOLD, LEAD, SILVER SULPHIDE, AND GOLD SULPHIDE IN EXTRA-SOLUTION.

§ 323. Solubility of Metallic Silver 46
§ 324. Solubility of Metallic Gold 47
§ 325. Solubility of Metallic Lead 47
§ 326. Solubility of Silver Sulphide 47
§ 327. Solubility of Gold Sulphide 48
§ 328. Solubility of Silver Arsenate and Antimonate 49

THE EFFECT OF EXTRA-SOLUTION ON SILVER-ORES.

§ 329. The Effect of Extra-Solution on Raw Ores 49
§ 330. The Effect of Extra-Solution on Ores that have been Oxidized by Roasting 54

THE EFFECT OF EXTRA-SOLUTION ON ORES THAT HAVE BEEN SUBJECTED TO A CHLORIDIZING-ROASTING.

§ 331. A. Roasted Ores free from Caustic Lime 59
§ 332. B. Roasted Ores containing Caustic Lime 61
§ 333. The Effect of Extra-Solution on Tailings 62

CHAPTER IV.

THE EXTRACTION OF GOLD 63

CHAPTER V.

CHEMISTRY OF THE WASH-WATER.

§ 501. The Composition of Roasted Ores 64
§ 502. Necessity of Leaching 64

A. Salts Easily Soluble in Water.

Page.
§ 503. Solubility of Sodium Sulphate 65
§ 504. Solubility of Sodium Chloride 65
§ 505. Solubility of Calcium Chloride 65
§ 506. Solubility of the Sulphates and Chlorides of Manganese, Zinc, Copper, ($CuCl_2$) Iron, Aluminium, and Magnesium. 65

B. Salts not freely Soluble in Water.

§ 507. Solubility of Cuprous Chloride 66
§ 508. " " Lead Chloride 66
§ 509. " " Calcium Sulphate 66
§ 510. " " Caustic Lime 66
§ 511. " " Sodium Antimonate 67

C. Salts not easily soluble in Water, but more soluble in Brine.

§ 512. Solubility of Cuprous Chloride 67
§ 513. " " Lead Chloride 67
§ 514. " " Calcium Sulphate 67

D. Salts practically insoluble in Water, but soluble in Brine, and in Solutions of other Chlorides.

§ 515. Solubility of Silver Chloride in Brine 68
§ 516. Solubility of Silver Chloride in Solutions of various Chlorides 69

E. Salts insoluble in Water, but soluble in Leaching-Brine.

§ 517. Solubility of Silver Chloride 71
§ 518. " " Silver Arsenate and Antimonate 71
§ 519. " " Lead Sulphate 71
§ 520. " " Lead Antimonate 72
§ 521. Conclusions 72

CHAPTER VI.

The Chemistry of Sodium and Calcium Sulphide.

A. Preparation of Sodium and Calcium Sulphide Solutions, and their Composition.

§ 601. Mode of Preparation 74
§ 602. Formation of Hyposulphite Salts 75
§ 603. Economy of producing Hyposulphites by Oxidation of Sulphide Solutions 76

Page.
§ 604. Circumstances influencing the Value of the Precipitating-Coefficients of Sodium and Calcium Sulphide 76
§ 605. Theory of high Precipitating-Coefficients 78
§ 606. Economy of using Sodium Sulphide with high Precipitating-Coefficient 80
§ 607. Sodium Monosulphide 81
Precautions in preparing Sodium Sulphide.
§ 608. Influence of diluting a Sodium Sulphide Solution upon the Value of its Precipitating-Coefficient 82
§ 609. Decomposition of Sodium Hyposulphite by Heating 83
§ 610. Precipitating-Coefficients of Sodium Sulphide for Lead and Copper 83
§ 611. Further Examination regarding the Effect of diluting a Sodium Sulphide Solution 84
§ 612. Summary 85

B. Calcium and Sodium Sulphide compared as Precipitants for Silver.

§ 613. The Cost of precipitating with Calcium and Sodium Sulphides at the Cusihuiriachic Mill 86
§ 614. Calcium and Sodium Sulphide compared as Precipitants, mechanically speaking 87
§ 615. Precipitation of Gypsum by Calcium Sulphide 87
§ 616. The Deterioration of a Lixiviation-Solution after Prolonged Use 88
§ 617. The Solvent Energy of various Hyposulphite Solutions for Silver Chloride 89
§ 618. The Dissolving Energy of Sodium and Calcium Hyposulphite Solutions for Gold 89

CHAPTER VII.

Laboratory Work.

§ 701. Determination of the Percentage of Salt in Crushed Ore. 92
§ 702. Determination of Soluble Salts in Roasted Ore 92
§ 703. Lixiviation-Test with Ordinary Solution 93
§ 704. Influence of repulverizing Roasted Ore in an Iron Mortar. 93
§ 705. Influence of Copper Vessel 94

LIXIVIATION-TESTS WITH EXTRA-SOLUTION.

Page.
§ 706. Russell's Selection of Lixiviation-Tests with Extra-Solution Criticised 96
§ 707. Control of the Mill-Work by Laboratory-Tests essential in Lixiviation 97
§ 708. General Remarks regarding the Lixiviation-Tests with Extra-Solution 98
§ 709. Lixiviation-Tests with Extra-Solution for Roasted Ores .. 98
§ 710. Lixiviation-Tests with Extra-Solution for Raw Ores 101
§ 711. Volumetric Determination of Sodium Hyposulphite with Iodine 102
§ 712. Influence of Silver, Copper, and Lead on the Correctness of Results 103
§ 713. Assaying the Sulphides for Silver 104
§ 714. Assaying the Sulphides for Gold 106
§ 715. Testing Ores regarding their Fitness for Lixiviation 106

PART II.

THE LIXIVIATION-PROCESS IN ITS PRACTICAL EXECUTION.

CHAPTER XI.

THE LIXIVIATION-PLANT.

Dimensions.—Under-Structure of Vats and Tanks.

§ 1101. Storage-Tanks for Stock-Solution, and Extra-Solution Tank 112
Dimensions.—Coil for heating Solutions.—Tank-Connections.—Chemical Boxes.—Outlets.—Korting Ejector.—Pipes for drawing off Sediment.—Solution-Pipes.—Valves.
§ 1102. Lixiviation-Vats 114
Advantages of Large Size.—Calculation of Capacity.—Lumber for Vats.—The Staves and Bottoms.—Filters.—Hoops.—Troughs for Charging Solutions.—Solution-Outlets, and Korting Ejector.—Triple Trough in front of Vats.—Tailings Shoot.—Sluice-Gate.—Painting the Vats.—Vats of Cement-Concrete.
§ 1103. The Precipitating-Tanks 119
Dimensions.—Outlets for Solutions.—Outlets for Precipitates.—Oars for Stirring.—Mechanical Stirrers.—Trough for Solution and Wash-Water.—Trough for Sulphides.—Precipitating-Tanks for Lead.

Page.
§ 1104. Storage-Tank for Sulphides 121
§ 1105. The Solution-Sumps 121
§ 1106. The Sodium Sulphide Tanks 122
Pipe-Lines.
§ 1107. Tanks for Sodium Carbonate Solution 123
§ 1108. The Korting Ejector 123
§ 1109. Pumps for Raising the Stock-Solution to the Storage-Tanks and for Sluicing the Tailings 125
§ 1110. The Filter-Press and the Pressure-Tank 126
The Filter-Press.—The Pressure-Tank.
§ 1111. The Drying-Chamber for Sulphides 129
§ 1112. Steam Boiler 129
§ 1113. The Arrangement of the Lixiviation-Plant 132
Index to Parts of Plant in Drawings, Plates I. and II.

CHAPTER XII.

The Solutions.

§ 1201. Definition of Terms 134
Acid Ore.—Alkaline Ore.—Arsenical Alkaline Ore.—Circulating.—Volume of Solution to Saturate a Charge.
§ 1202. The Ordinary or Stock-Solution 134
Preparation.—Concentration in Sodium Hyposulphite.—Volumes of Stock-Solution used per ton of Ore.—Change in Concentration by Continued Use.—Loss in Sodium Hyposulphite.—Gain in Sodium Hyposulphite.—Maintenance in Strength.—Determination of Concentration.—Concentration in other Salts.—Caustic Stock-Solution.—Volume of the Stock-Solution in the Mill.
§ 1203. The Extra-Solution 139
Extra-Solution not kept in Stock.—Composition of Extra-Solution.—Rules for Preparing Standard Extra-Solution.—Modes of Preparation: First Method.—Testing Standard Extra-Solution.—The Second Method.—Percentage of Copper Sulphate Used.—Volume.—Depth.—Circulation.—Effect of One Pound Copper Sulphate.—Temperature.
§ 1204. The Sodium Carbonate Solution 142
§ 1205. The Sodium Sulphide Solution 142
Manufacture.—Dilution.
§ 1206. The Calcium Sulphide Solution 144

CHAPTER XIII.

THE CHARGING AND DISCHARGING OF THE LIXIVIATION-VATS.—THE SECOND WASH-WATER.—GENERAL RULES FOR HANDLING THE LIXIVIATION-SOLUTIONS.

Page.

§ 1301. Charging the Lixiviation-Vats with Ore 145

Charging by Cars.—Charge the Ore Dry.—Charge the Ore Cold.—Weight of the Charge.—Filling the Vats.—Shrinkage of the Charge.—Precautions.

§ 1302. General Rules for Handling the Lixiviation-Solutions .. 147

Volume to Saturate.—First Wash-Water.—Lixiviation.—Extra-Solution.—Second Wash-Water.—Measurement of all Solutions.—The Rate of Lixiviation.—Influence of Depth.—Influence of Fine Crushing.—Increasing the Rate of Lixiviation by the Korting Ejector.—Irregular Shrinkage of the Charge.

§ 1303. The Second Wash-Water 151

§ 1304. Discharging the Tailings and their Sampling 152

Sampling the Tailings.—Silver Sulphide on the Surface of the Charge.

CHAPTER XIV.

THE TREATMENT OF ROASTED AND RAW ORES.

§ 1401. The First Wash-Water 154

Leaching from Above.—Leaching from Below.—Quantity of First Wash-Water.—Displacing the First Wash-Water.—Leaching with Cold or Hot Water.—Distribution of Silver in Wash-Water.—Silver extracted by the First Wash-Water from various Ores.

TREATMENT OF THE SILVER-BEARING WASH-WATER.

§ 1402. Extraction of Silver from Wash-Water by Dilution with Cold Water 158

§ 1403. Extraction of Silver from Wash-Water by Precipitation with Sodium or Calcium Sulphide 158

§ 1404. Extraction of Silver from Wash-Water by Precipitation with Scrap-Iron 159

THE LIXIVIATION.

§ 1405. Lixiviation of Raw Ores 160

Explanation of Table No. 1405.

§ 1406. Lixiviation of Acid Roasted Ores 161

Effect of Circulation.

§ 1407. Lixiviation of Arsenical Alkaline Roasted Ores 163

Page.
§ 1408. Lixiviation of Alkaline Roasted Ores 163
§ 1409. Distribution of Silver and Lead in the Solution during different Periods of Lixiviation 165
§ 1410. Method of determining whether the Extraction of the Silver is completed or not 165

CHAPTER XV.

TROUGH-LIXIVIATION 167

CHAPTER XVI.

THE PRECIPITATION OF THE METALS FROM A LIXIVIATION-SOLUTION.

§ 1601. Constitution of a Lixiviation-Solution before Precipitation 171
§ 1602. The Precipitation of Lead with Sodium Carbonate 172
Manipulation.—Quantity of Lead dissolved in Lixiviation.—Economy of Precipitating Lead as Carbonate.—The Precipitation of Lead by Sodium Carbonate and Sulphide compared.
§ 1603. The Precipitation of Silver, Gold, and Copper 174
Manipulation.—Testing for the End of the Reaction.—Over-precipitation.—Precipitation of Calcium Sulphide.—Decanting the Solution.—Value of the Sulphides.
§ 1604. Mechanical Handling of the Sulphides 176
Manipulation of the Pressure-Tank and the Filter-Press.—Capacity of the Filter-Press.—Moisture in Sulphide-Cakes.—Disadvantages of leaving the Sulphides too long in the Precipitating-Tanks.
§ 1605. Drying of the Sulphides 177
§ 1606. Refining of the Sulphides 177
The Old Practice.—Crucible Melting at Cusihuiriachic.—Cupellation at Cusihuiriachic.—Loss of Silver in Roasting.—Refining by Humid Processes.—New Method Proposed.—The Roessler Convertor.—Advantages of the Process.
§ 1607. Selling Sulphides to Smelters 181

CHAPTER XVII.

The Construction, Cost of Erecting, and Expense of Running a Lixiviation-Mill.

A. Notes on Construction.

Page.
§ 1701. Capacity of Krom's Rolls and Stetefeldt Furnace 183
§ 1702. Arrangement of the Crushing-Plant 184
§ 1703. Rock-Breakers 184
§ 1704. Advantages of Perfect Drying 184
§ 1705. Details of Mill-Construction 185
Feeders for Rolls.—Elevators.—Shoots.—Hoppers.—Conveyors.—Pulp-Feeder.—Screen-Tower.—Speed of Engine and Line-Shaft.—Sellers' Hoist.

B. Estimated Cost of Mill-Plant.

§ 1706. Estimated Cost of Plant for Drying, Crushing, and Roasting the Ore 187
§ 1707. Estimated Cost of the Lixiviation-Plant Proper 189
§ 1708. Total Cost of Erection 192

D. Estimated Cost of Running a Lixiviation-Mill of 80 Tons Capacity per Day.

§ 1709. Labor 192
§ 1710. Fuel 193
§ 1711. Wear and Tear 194
§ 1712. Chemicals 194
§ 1713. Total Estimated Cost of Lixiviation 194
§ 1714. Cost of Chemicals in Stock 196

CHAPTER XVIII.

The Russell Process Compared with Ordinary Lixiviation, and with Amalgamation.

A. The Russell Process Compared with Ordinary Lixiviation.

a. Roasted Ores.

§ 1801. Results at the Ontario Mill, Park City, Utah, in 1883–1884 198
§ 1802. Results at the Sierra Grande Mill, Lake Valley, New Mexico 199

Page.
§ 1803. Results at the Sombrerete Mill, Zacatecas, Mexico 200
§ 1804. Results at the Cusihuiriachic Mill, Chihuahua, Mexico. 201
§ 1805. Results at the Yedras Mill, Sinaloa, Mexico 205

B. Raw Ores.

§ 1806. Results at the Flagler Reduction-Works, Silver City, New Mexico 210

B. The Russell Process Compared with Amalgamation.

A. Roasted Ores.

§ 1807. Results at the Ontario Mill, in 1883–1884 210
§ 1808. Results at the Marsac Mill, Park City, Utah, in 1887–1888 212
§ 1809. Results at the Ontario Mill in 1887–1888 213
§ 1810. Results at the Ontario Mill in 1888 214
§ 1811. Results at the Sierra Grande Mill, Lake Valley, New Mexico 219
§ 1812. Results at the Cusihuiriachic Mill, Chihuahua, Mexico .. 220

B. Raw Ores.

§ 1813. Results at the Big Sandy Mill, Arizona, comparing the Russell Process with Combined Raw Amalgamation and Concentration 221
§ 1814. Relation of Mill-Results by Amalgamation and by the Russell Process, to Results respectively obtained in the Assay-Office with Ordinary and Extra-Solution .. 222
Index 225
Index to Drawings in Text 234
Advertisements 235

THE LIXIVIATION OF SILVER-ORES

WITH

HYPOSULPHITE SOLUTIONS,

WITH SPECIAL REFERENCE TO THE

RUSSELL PROCESS.

INTRODUCTION.

THE extraction of silver by the lixiviation-process from ores which have been subjected to a chloridizing-roasting, is based upon the fact that silver chloride is easily soluble in solutions of sodium or calcium hyposulphite, and that silver is precipitated from such solutions by an alkaline sulphide, with regeneration of the hyposulphite salts. In case the ore contains lead, a portion of the latter is also dissolved, lead sulphate being soluble in hyposulphite solutions. If, at the same time, copper is present in the roasted ore in the form of cuprous chloride, the sulphides precipitated from the lixiviation-solution contain silver, copper, and lead. From gold-bearing silver-ores, gold is obtained together with the silver, but the percentage of its extraction varies, and depends upon many circumstances. Only ores of low value in gold, say below one ounce per ton, yield a high percentage of this metal.

§ 1. *History of the Lixiviation-Process.*

This process was first introduced by von Patera, an Austrian metallurgist, at Joachimsthal, in 1858. Von Patera used sodium hyposulphite, for which Kiss substituted the calcium-salt, in 1860, at Schmoellnitz. In the United States, the process was first employed by Kuestel, in 1874, at Melrose, California, and later,

principally by O. Hofmann, it was introduced in several reduction-works in California, Nevada, and Arizona, none of which are, however, in operation at present. The use of the process was revived by the construction of the Bertrand Mill, in 1882, the Mount Cory Mill, in 1883, and the Cortez Mill, in 1885, all in Nevada.

In Mexico, lixiviation has obtained, since 1868, a much stronger footing than in the United States, and bids fair to supersede in the future pan-amalgamation altogether, if not the Patio process.

§ 2. *Defects of the Old Lixiviation-Process.*

Although lixiviation is much cheaper than amalgamation, both as to cost of plant and working-expense per ton of ore, there were many reasons why its adoption made slow progress. Leaving out the fact that there always exists a prejudice against any new method that requires more skill and chemical knowledge than those in vogue, the lixiviation-process, as executed by von Patera, Kiss, Kuestel, and Hofmann, had some serious defects which made it inapplicable to the treatment of certain ores. In the first place, it was necessary to subject the ore to a very perfect chloridizing-roasting. A high chlorination of the silver cannot always be obtained, especially in case the ore contains calc-spar, which is converted by roasting, in part, into caustic lime. The caustic lime not only reduces silver chloride to metallic silver, but also greatly diminishes the solubility of most silver-compounds in hyposulphite solutions. In the second place, if the ore contains lead and copper, these metals go, as already pointed out, in part, in solution together with the silver, and are, in the following operation, precipitated with the latter as sulphides. It is well known that such a combination of metals is not desirable for subsequent treatment. No difficulty, however, is experienced if copper only is precipitated with the silver. Finally, we have to consider that although a sodium or calcium hyposulphite solution dissolves—besides silver chloride—silver antimonate and arsenate, and, more or less, metallic or native silver, it does not attack at all either silver sulphide or silver-glance, or the group of precious silver minerals known as antimonial and arsenical sulphides, like polybasite, stephanite, ruby-silver, and fahl-ore. Hence, lixiviation of

some classes of raw ores did not succeed where raw amalgamation with chemicals was conducted with profit.

§ 3. *The Russell Process.*

These defects have been overcome by the Russell Process, so named after its inventor, Mr. E. H. Russell.

Mr. Russell discovered that a solution of a double-salt of cuprous hyposulphite and sodium hyposulphite, formed by mixing sodium hyposulphite with copper sulphate, exerts a most energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver-minerals belonging to the group of antimonial and arsenical sulphides, and other silver combinations. Hence, if a charge of roasted ore is first lixiviated with ordinary sodium hyposulphite solution to dissolve the silver chloride, and, subsequently, with cuprous hyposulphite,—this solvent is called the extra-solution,—an additional amount of silver is extracted which would have been lost in the tailings by working according to the old method alone. Or if the roasted ore contains caustic lime, and is treated with the extra-solution, the deleterious influence of the caustic lime is thereby counteracted. In the same way the extra-solution may be applied to extract silver from raw ores, without previous chloridizing-roasting, or to lixiviate ores after they have been subjected to an oxidizing-roasting.

Mr. Russell also discovered that lead can be completely separated from a sodium hyposulphite solution, as lead carbonate, by sodium carbonate, without precipitating copper or silver. After decanting the solution from the lead carbonate, silver and copper are obtained from it in the usual way. This method of separating lead prohibits the use of calcium poly-sulphide as a precipitant for the sulphides, because calcium entering the regenerated lixiviation-solution would also be precipitated as a carbonate with the lead by sodium carbonate. Hence, sodium sulphide must be employed. A full investigation has demonstrated that this is by no means detrimental. Sodium sulphide and hyposulphite are more advantageously used in the lixiviation-process than the corresponding calcium-salts.

Finally, Mr. Russell found that if a hyposulphite solution has a caustic reaction, produced by caustic soda or lime, its solvent

power for silver is materially deteriorated. This defect he corrects by neutralizing such a solution with sulphuric acid.

§ 4. *Differences between the Russell Process and Ordinary Lixiviation.*

The Russell process differs from the old or ordinary lixiviation-process in the following points:

1st. It requires a less careful chloridizing-roasting, and, on that account, a lower percentage of salt may be used in roasting.

2d. It extracts a higher percentage of silver by means of the extra-solution. This is especially of importance in treating ores raw, and in lixiviating roasted ores containing caustic lime.

3d. It produces sulphides free from lead.

4th. It yields lead in the form of lead carbonate as a valuable by-product.

5th. It overcomes the deleterious effect of a caustic lixiviation-solution by neutralizing it with sulphuric acid.

6th. It uses sodium hyposulphite and sulphide exclusively, and not the corresponding calcium-salts.

§ 5. *The Russell Process compared with Amalgamation.*

In comparing the Russell process with amalgamation, the principal items in favor of lixiviation are:

1st. In amalgamation, the coarseness of crushing, without considering the question of roasting, is limited by the capacity of the settler to work off coarse sands without loss of quicksilver. In lixiviation, pulverizing as coarse as possible is desirable. The limit of coarseness depends upon the character of the ore, and, principally, upon the manner in which the silver-bearing minerals are distributed in the gangue.

2d. The original cost of the lixiviation-plant is much lower than that of pans and settlers. A further saving is effected by a reduction in size of the engine and boilers.

3d. In amalgamation the pans and settlers consume not less than $1\frac{1}{8}$ horse-power per ton of ore. The power for pumping solutions, etc., in the lixiviation-process, is merely nominal.

4th. In large mills the quantity of quicksilver in rotation repre-

sents a capital of from $30,000 to $40,000, while the stock of chemicals required for lixiviation costs less than one-tenth of this amount.

5th. With Russell's improvements, the percentage of silver extracted by lixiviation is in most cases higher than by amalgamation.

6th. Lixiviation by Russell's process requires a less careful chloridizing-roasting, and, in consequence, a lower percentage of salt may be used in roasting.

7th. Ores that can be successfully treated by raw amalgamation give often better results by raw lixiviation with extra-solution.

8th. The value of the lost quicksilver, and cost in wear and tear of the pans and settlers, amounts to more than that of the chemicals consumed in the lixiviation-process.

9th. The lixiviation process permits the extraction of copper and lead as valuable by-products.

10th. Amalgamation is invariably injurious to the laborer's health.

11th. In case gold-bearing silver-ores have been roasted with salt, lixiviation extracts, in many cases, more gold than amalgamation.

The disadvantages of lixiviation, as compared with amalgamation, are:

1st. Lixiviation requires more chemical knowledge, and a more careful supervision of the operations.

2d. The handling of large volumes of solutions is a disadvantage common to all humid processes.

3d. There is more danger of losing silver by careless manipulation, and by leakage of badly constructed plant.

4th. In the lixiviation-process the precious metals are obtained in the form of sulphides. The conversion of the latter into bullion requires more skill, and is more expensive than the handling of amalgam.

§ 6. *History of the Russell Process.*

The great superiority of the Russell process, as compared with ordinary lixiviation and with amalgamation, has given a new impetus to the introduction of lixiviation, especially in Mexico.

The Russell process was first introduced in September, 1884, at

Silver Reef, Utah, for the raw lixiviation of tailings, resulting from raw amalgamation of the silver-bearing sandstones, peculiar to this locality.

In December, 1884, Mr. Russell attempted the raw lixiviation of tailings, at Bullionville, Nevada. These tailings—the result of raw amalgamation of oxidized ores from the mines of Pioche District, Nevada—had been re-worked by amalgamation, and also by concentration. Their contents in precious metals were from 8 to 12 ounces silver, and about $2.50 gold per ton. A more unfavorable material, especially in a mechanical sense, could not be selected, since repeated grinding in pans had reduced the ore almost completely to slimes. The result, both as to extraction of silver and manipulation, was a complete failure. It would not be of interest to point out all the difficulties encountered. After roasting the tailings in a Stetefeldt furnace, they lixiviated well, mechanically speaking; but here another obstacle was met. The composition of the tailings was such that without the admixture of sulphuret-ores, high chlorinations of the silver could not be obtained; and the extra-solution did not correct this defect in roasting sufficiently. The failure of procuring sulphuret-ores at a reasonable cost; the expense of drying the tailings; the high price of fuel, salt, and labor at Bullionville, left no profit, and caused the closing of the mill. I consider it of sufficient importance to point out these facts, because the disastrous results of the Bullionville enterprise created the impression in many quarters that the Russell process itself was a failure.

Somewhat better were the results in lixiviating tailings at the Flagler Reduction-Works, Silver City, New Mexico, in 1885. Here also tailings from raw amalgamation, mostly in the form of slimes, made filtering exceedingly slow and difficult. It should be noted that, in both cases, the Russell process was applied, as a last resort, to worthless material from which silver could not be extracted with profit by any other process.

In February, 1886, the Sierra Grande Co. put into operation a large lixiviation-plant, calculated to work 80 tons of ore per day. These works, situated at Lake Valley, New Mexico, were designed to treat ores after they had been subjected to a chloridizing-roasting. The short-sighted policy of the company caused the erection of a cheap and badly constructed plant, designed and built by

inexperienced engineers. The mill, mechanically speaking, was a complete failure; the success of the Russell process, however, was fully established. The ore was of such a character that ordinary lixiviation extracted the silver very imperfectly. On account of the expense of running a badly constructed mill, the machinery of which was constantly breaking down, the low-grade ore extracted from the mines did not yield a profit, and operations were soon suspended, the company being without funds to reconstruct the mill.

At the Ontario Mill, Park City, Utah, Mr. Russell experimented on a large scale in 1883–84. The results of these experiments are recorded in § 1801 and § 1807. Mr. W. A. Wilson commenced a new series of lixiviation-tests with Ontario and Daly ore in 1887, as will be seen in § 1808 and § 1809. These experiments are now continued by Mr. Russell, and on their success depends whether lixiviation will take the place of amalgamation in the Ontario Mill. The ore now extracted from the Ontario mine is very different in character, as compared with ore mined in 1883–84. With very high chlorination of the silver, the percentage extracted by amalgamation from roasted ore, falls short of the amount of silver extracted by ordinary solution in the assay-office; hence, the ore is more suitable to treatment by lixiviation than by amalgamation, and seems especially adapted to treatment by the Russell process. Better opportunities were offered to the Russell process in Mexico.

Mr. Ellsworth Daggett, General Manager of the Casihuiriachic Co., in the State of Chihuahua, adopted the Russull process in 1886. After its usefulness there had been fully established, by repeated careful investigations regarding its superiority, as compared with ordinary lixiviation and pan-amalgamation, a new manager succeeded Mr. Daggett, in 1888, whose first step was to abandon the Russell process, returning to the old mode of working, for reasons not stated.

A brilliant success is recorded in the Yedras Mill, Sinaloa, where the Russell process was substituted, in November, 1887, for ordinary lixiviation. The new Sombrerete Mill, in the State of Zacatecas, will also use the Russell process, and the plant, now in course of construction, is especially designed for this purpose. Tailings from ordinary lixiviation, and from amalgamation, were treated with profit at Parral, by Mr. Koehler.

§ 7. *Ores Suitable to Treatment by Lixiviation.*

Almost all silver-ores that do not carry a large percentage of lead or copper can be treated by lixiviation with success and economy. I do not mean to create the impression that from ores containing more or less lead and copper a high percentage of the silver can not be extracted by this process. Such ores, however, will, in most localities, be reduced to better advantage by smelting.

It is safe to state that all ores fit for amalgamation can also be treated by lixiviation, and that the Russell process may succeed where amalgamation is a failure. The exceptions to this rule will be rare. In case lead-bearing silver-ores are suitable for concentration, it may be profitable to concentrate the ore by Krom's dry system, obtaining a smelting-product high in lead, and to lixiviate, after roasting, the tailing and the dust.

The dry system of concentration deserves the preference, because it delivers the tailings and the dust in a condition ready for chloridizing-roasting. In wet concentration the drying of the tailings would be expensive, and there would be a considerable loss of siver in slimes. Oxidized ores, containing silver chloride and lead-minerals, may be lixiviated after crushing, and the tailings may be concentrated for lead. In this case wet concentration would be most suitable. This has been done at the Old Telegraph Mine, Utah.

The Russell process is also adapted to the treatment of tailings resulting from ores which have been worked either by the old lixiviation-process or by amalgamation. Whether it is most profitable to lixiviate an ore raw, or after chloridizing-roasting, or after oxidizing-roasting, must be determined in each case by actual experiment.

§ 8. *The Operation Preceding Lixiviation.*

A detailed description of the operations preceding the lixiviation-process, namely: crushing, or crushing and roasting of the ore, is not necessary here, since these operations are identically the same as those preceding the well-known treatment of amalgamation. In every instance the crushing should be done dry, even in case the ore is lixiviated raw.

As to crushing-machinery, I would always give preference to well constructed rolls, like those furnished by Mr. S. R. Krom, in place of stamps. Pulverizing by rolls produces much less fine dust, and a minimum quantity of the latter facilitates the filtering of the solutions through the charge in the lixiviation-vats. The size or number of the screen through which the crushed ore should be made to pass, in order to lixiviate with most economical results, can only be determined by a practical test in each individual case, since it depends upon the character of the ore. A No. 8 wire-screen may generally be considered as the limit of coarseness. In many cases a No. 20 screen will be used to best advantage, especially if the ore has to be roasted. A screen, finer than No. 30, will rarely be needed. (These commercial designations correspond with the number of meshes per linear inch.)

All this refers to crushing by rolls, and screening through revolving screens. In dry-crushing pan-mills, where the ore is subjected to a chloridizing-roasting, only stamps have been used so far. In such mills screens, not coarser than No. 30, are generally used on the battery, the settlers being unable to do good work with coarser material. Stamp-crushing through a No. 30 screen is about equivalent to crushing by rolls, and screening through a revolving screen with No. 40 wire-cloth.

At the Ontario Mill, No. 26 screens are used without detriment to the working of the settlers, and at the Daly Mill (both in Park City, Utah), No. 20 screens have been successfully introduced by Mr. W. A. Wilson. These, however, are exceptions to the general rule.

In roasting the ore, those furnaces should be used in which the dust collected in the dust-chambers is well chloridized. The Stetefeldt furnace is known to give the most perfect results in that respect. In case the dust remains raw, the proportion of silver extracted is not only diminished for chemical reasons, but the mechanical difficulties in lixiviating such fine material are also increased in proportion to the imperfection of its roasting.

A further advantage of the instantaneous roasting in a Stetefeldt furnace is that it leaves the silver-compounds in a more soluble state than if the ore has been roasted in other furnaces, where it remains exposed to the action of the flame for a long time. This becomes especially apparent in the oxidizing-roasting of sulphuret-ores, as will be demonstrated later.

§ 9. *Effect of Wetting Down Roasted Ore while Red-Hot.*

In wetting down roasted ore on the cooling-floor, a very injurious effect may be produced if this operation is performed while the ore is red-hot. The effect is most pronounced in case ordinary lixiviation is used, but a considerable difference also exists in treating the ore with extra-solution. Evidently the steam, coming in contact with the ore at a high temperature, causes a decomposition of certain silver-compounds, otherwise soluble in hyposulphite solutions. The following table will illustrate this fact:

TABLE No. 9.

SHOWING THE EFFECT OF WETTING DOWN ROASTED ORE WHILE RED-HOT.

NAME OF MILL.	Value of Ore per ton. Ounces.	Extraction of Silver with			
		Ordinary Solution, Wetting Down.		Extra-Solution. Wetting Down.	
		Cold. per cent.	Red-Hot. per cent.	Cold. per cent.	Red-Hot. per cent.
Cusihuiriachic,	49.1	69.1	20.3	86.3	80.3
Sierra Grande,	16.0	81.5	62.5	87.5	71.9

The extraction in the mill was approximately the same as that indicated by the lixiviation-tests in the assay-office.

§ 10. *The Stetefeldt Furnace.*

Finally, I return to the question of choice of roasting furnace. Since the patents of the Stetefeldt furnace have expired several years ago, no bias can be imputed to my opinion, if I recommend this furnace as the most advantageous one to prepare ore for lixiviation. This subject has been fully treated by Mr. Ellsworth Daggett in his paper on the Russell process, published in Vol. XVI of the Transactions of the American Institute of Mining Engineers, and he fully endorses my opinion. The large capacity of this furnace, its durability, economy in fuel and labor, adaptability to treat all classes of ores, the perfection with which it roasts the dust, minimum loss of silver in roasting by volatilization, and perfect precipitation of the dust in well-constructed chambers, make it preëminently the choice for works of large capacity. At the Ontario Mill the Howell furnace would not have been discarded but for good reasons, and at the Daly Mill, with two Howell furnaces on hand, the Stetefeldt furnace would not have been

built unless its superiority had been fully demonstrated. Also the Blue Bird Mill, Butte, Montana, decided to substitute the Stetefeldt furnace for the Howell, the latter having already been ordered. Of other mechanical roasting furnaces in vogue, there remains but the Brueckner Cylinder. While this furnace has given excellent results in many mills, it has been a complete failure in treating certain classes of ores, of which the Yedras ore is a prominent example. For details on this subject I refer to Mr. Daggett's paper and to Mr. George J. Rockwell's articles, "Chloridizing-Roasting and Lixiviation at Yedras Mine," published in the Eng. and Mg. Journal, Feb. 4th, 1888, and following numbers.

PART I.

THE CHEMISTRY

OF THE

LIXIVIATION-PROCESS.

CHAPTER I.

The Chemicals.

The chemicals used in the lixiviation-process are: Sodium hyposulphite, for making the stock-solution; copper sulphate, for preparing Russell's extra-solution; sodium carbonate, for precipitating lead; caustic soda, caustic lime, and sulphur, for preparing sodium or calcium sulphide; sulphuric acid, for neutralizing caustic lixiviation-solutions, and for treatment of the wash-water.

§ 101. *Sodium Hyposulphite.*

Sodium hyposulphite is the only hyposulphite salt that is manufactured on a large scale. It is principally imported from England and Germany. It has the formula $Na_2S_2O_3+5$ aq., and does not deteriorate in contact with the atmosphere. Its solutions are also quite permanent, and decompose slowly either at ordinary temperature or at boiling-point.

The solubility of sodium hyposulphite is as follows. 1000 c. c. water dissolve:

At 0° C., or 32° F.,	497·5 gm.	$Na_2S_2O_3+5$ aq.
" 20° C., or 68° F.,	694·4 "	" "
" 40° C., or 104° F.,	1041·6 "	" "
" 60° C., or 140° F.,	1923·0 "	" "

Acids added to the solution precipitate sulphur, and evolve sulphurous acid. The crystals of $Na_2S_2O_3+5$ aq. melt at 56° C., or

132.8° F.; at 100° C., or 212° F., the water is completely driven out. Upon heating the dry salt, it emits sulphur, and a yellow mass remains, consisting of sodium sulphide and sulphate.

§ 102. *Calcium Hyposulphite.*

Calcium hyposulphite, CaS_2O_3+6 aq., is not an article of commerce. It can only be made with difficulty in the laboratory, since its solution has to be evaporated at a temperature below 60° C., or 140° F. At a higher temperature decomposition takes place, sulphur and gypsum being formed. Calcium hyposulphite solutions also deteriorate much quicker than those of the sodium-salt at ordinary temperature. Lixiviation on a large scale is generally commenced with a sodium hyposulphite solution. If then calcium sulphide is used as a precipitant for silver, the sodium hyposulphite is gradually replaced by the calcium-salt. Under all circumstances it is recommended to start lixiviation with sodium hyposulphite, and not to manufacture calcium hyposulphite at the reduction-works, as has been proposed, by conducting sulphurous acid through a solution of calcium sulphide until the latter is completely decomposed. Since sodium hyposulphite can be bought so cheaply, there is not even economy in the rather tedious manufacture of the calcium-salt.

§ 103. *Sodium Carbonate.*

It is most economical to buy the soda made by the Solvay or ammonia process, and not the impure common soda-ash. The Solvay soda is now manufactured in this country, at Syracuse, N. Y.

This alkali, in the form of a white powder, is so pure that it contains 98.7 per cent. of sodium carbonate, the remainder being principally sodium chloride and sulphate, and 0.25 per cent. moisture. On account of the entire absence of sodium sulphide, (which is always found in the ordinary soda-ash), the solution for the precipitation of lead does not require purification.

The solubility of sodium carbonate is as follows. 1000 c. c. water dissolve:

At 0° C., or 32° F., 69.7 gm. Na_2CO_3.
" 10° C., or 50° F., 120.9 " "
" 20° C., or 68° F., 217.1 " "

§ 104. *Caustic Soda.*

Only a high grade article should be ordered, of 70 to 76 per cent. It is put up in sheet-iron drums, holding 630 to 700 pounds. It should not be left exposed to the air, after a drum has been opened, as it absorbs carbonic acid.

The solubility of caustic soda is as follows. 1000 c. c. water dissolve:

At 15° C., or 59° F., 600 gm. NaHO.
" 100° C., or 212° F., 2100 gm. NaHO.

A saturated solution of caustic soda boils at 215.5° C., or 330° F. Caustic soda of high grade is manufactured by the Solvay Process Co., Syracuse, N. Y.

§ 105. *Commercial Rating of Soda and Caustic Soda.*

The manufacturer of soda-ash and soda, rate their articles in a different way in different countries. In Germany, the percentage in Na_2CO_3 is given; in France, the percentage in Na_2O, and England follows the French custom, but substitutes 24 for the equivalent of Na, which is 23. This is one of the numerous mysteries and humbugs of English commerce. Hence, the equivalent of Na_2O is taken at 64, and not at its true value of 62. The equivalent of Na_2CO_3 is taken at 108, in place of 106. Chemically pure soda, free from water, would be rated in Germany, at 100, in France, at 58.5, and in England, at 59.27 per cent. One French Gay-Lussac per cent. = 1.013 English per cent.

The Solvay soda is generally rated at 58 per cent., and is consequently almost pure Na_2CO_3. The quotations of prices here, and in England, are based upon an article of 48 per cent., and purer qualities cost more in proportion to their percentage of Na_2O above 48 per cent. In regard to the valuation of soda-ash, it is necessary to mention that its valuation also includes the free NaHO, generally contained in it. The Germans calculate this free NaHO as Na_2CO_3, so that a soda-ash may be quoted, in Germany, as containing more than 100 per cent. Na_2CO_3.

As to caustic soda, not its percentage in NaHO, but in Na_2O is given, as is the case with soda in France and England, and what has been stated about the change of the equivalent of Na in England, applies here also. Pure NaHO contains 77.5 per cent.

Na_2O, but, according to false English conceptions, 78.04 per cent. Na_2O. Quotations of caustic soda are based upon an article of 60 per cent. I give below analysis of some samples of English caustic soda.

Analysis of Caustic Soda.

NaHO	87.73 per cent.	87.50 per cent.	85.71 per cent.
Na_2CO_3	3.45 "	3·68 "	5.48 "
Na_2SO_4	1.00 "	1.32 "	2.10 "
NaCl	7.23 "	6.89 "	5.60 "
H_2O	0.59 "	0.68 "	1.11 "

These samples would be rated at 68, 67.8, and 66.4 per cent., according to French and German standards; and at 68.9, 68.7, and 67.3 per cent., respectively, according to English custom. The best qualities of caustic soda in the market generally contain from 70 to 74 per cent. Na_2O, and none is manufactured above 76 per cent.

§ 106. *Sulphur.**

It is immaterial whether the sulphur is crude or refined, in lumps, or flowers of sulphur. In the preparation of sodium sulphide, fine pulverizing of the sulphur is not necessary.

§ 107. *Sulphuric Acid.*

Sulphuric acid of 66° B. is best transported in sheet-iron tanks which hold about 1700 pounds. If shipped in this way, the freight is much less than in glass carboys. The cost of a sheet-iron tank is $6.50.

§ 108. *Prices of Chemicals.*

The prices of these chemicals, especially that of copper sulphate, vary like those of all commercial articles; hence, quotations would be of little value. I will state, however, that sodium hyposulphite, if imported in large lots from Europe, costs, in New York, about $1\frac{9}{10}$ to 2 cents per pound.† The price of a 74 per

* In the West, sulphur can be bought from the Dickert & Myers Sulphur Co., Salt Lake City, Utah.

† This refers to an article of best quality, imported from Germany. Sodium hyposulphite of English manufacture. and of low quality, sells for less. Only the best article should be bought for lixiviation.

cent. caustic soda is, in New York, about 3 cents per pound, and that of Solvay soda about $1\frac{4}{10}$ cents per pound.

The Chemical Reactions.

The chemistry of the lixiviation-process, simple as it may appear at first glance, is by no means fully explored, as will be seen in subsequent chapters of this treatise. Much valuable work has been done, but many phenomena still remain unexplained, and will require delicate chemical research. In the following pages I have endeavored to give a concise statement of the reactions that are of practical value. Those who wish to put lixiviation-works into successful operation should make themselves thoroughly familiar with the facts recorded below. We have to deal here, principally, with questions of solubility.

The solubility of a substance in a liquid may be considered as a function of temperature, concentration of the liquid, and time of reaction, other conditions being equal. In regard to the solubility of certain combinations of silver, and other metals, in solutions of sodium hyposulphite, we may state here at once that temperature enters in so far as an important factor, as moderately warm solutions, say from 25° to 55° C., or 77° to 131° F., act more energetically, and dissolve in less time, than colder ones. A much higher temperature of the solution, however, should be avoided.

In some of the following investigations on solubility it might have been of interest if, in every case, a series of experiments had been conducted with the view to arrive quickly and surely at saturation-points by using less solution, and a larger surplus of the substance to be dissolved. I do not think, however, that my general conclusions, drawn from these researches, would be affected by such additional information. This has been verified in special cases. I also wish to say, that if others should repeat similar experiments, under somewhat different conditions, they should not be astonished at finding numerical results differing, more or less, from those recorded here. They will, undoubtedly, arrive at the same general conclusions.

CHAPTER II.

The Reactions of Sodium Hyposulphite Solutions.

§ 201. *Argentic Hyposulphite Double-Salts.*

Silver forms with the hyposulphites of sodium, potassium, and calcium two series of double-salts. The one has the formula $2RS_2O_3+Ag_2S_2O_3+3aq$, and the other, $RS_2O_3+Ag_2S_2O_3+2aq$.

Of these double-salts the former are easily soluble in water, and the latter much less so.

Gold forms soluble double-salts of the formula: $3RS_2O_3+Au_2S_2O_3+4aq$.

The solutions of the argentic double-salts are quite permanent, and if neutral or alkaline, can be heated to boiling-point without injury. The addition of a very slight quantity of acid to a boiling solution, however, starts decomposition, silver being precipitated as sulphide. A slightly acid solution keeps at ordinary temperature; it commences to decompose if heated above 50° C., or 122° F.

The argentic hyposulphites differ materially in their reactions from other silver-salts. The most important differences are as follows, viz:

Chlorides, or hydrochloric acid, do not precipitate silver chloride.

Caustic soda does not precipitate silver oxide.

Sodium carbonate does not precipitate silver carbonate.

Solutions of potassium arsenate and antimonate do not precipitate silver arsenate and antimonate. The granular, white precipitate, which appears if potassium antimonate is added to the argentic sodium-salt, is sodium antimonate.

Ferrous sulphate does not precipitate metallic silver, either at ordinary temperature, or upon heating.

Metallic copper and lead precipitate the silver as completely as from silver nitrate, but the reaction is slower.

§ 202. *Double-Salts of Lead and Sodium Hyposulphite.*

Lead forms a sodium double-salt of the formula: $2Na_2S_2O_3$, PbS_2O_3, which is soluble in an excess of sodium hyposulphite. If this solution is heated to about 70° C., or 158° F., it turns

slightly dark. Upon boiling for some time, further decomposition does not seem to take place. A solution to which sulphuric acid has been added, becomes milky from separated sulphur, and turns dark at about 50° C., or 122° F. Upon boiling, decomposition takes place, lead sulphide being precipitated. Complete decomposition, however, is slow, lead being still found in solution after continued boiling.

Sulphuric acid and sulphates do not precipitate lead sulphate from solutions of lead hyposulphite.

In regard to caustics and carbonates of the alkalies, the reactions are the same as those of other lead-salts. The precipitation with sodium carbonate is very complete. Caustic soda precipitates a basic hydroxide of lead. The latter is easily soluble in caustic soda; hence, if an excess of the reagent is added, lead remains in solution. From the filtrate, sodium carbonate does not precipitate lead carbonate, the latter being also soluble in caustic soda.

The characteristics of the cuprous hyposulphites will be discussed in the next chapter.

Solubility of Metals and Various Compounds in Sodium Hyposulphite Solutions.

§ 203. *Solubility of Metallic Silver.*

[In all the following experiments the metric system is used.]

0.500 gm. cement-silver were treated cold (at about 50° C., or 59° F.) for forty-eight hours with 1000 c. c. of solutions, containing respectively 10, 25, 50, 100, 150, 200, and 250 gm. $Na_2S_2O_3+5aq$. The result was that the 1 per cent. solution dissolved as much silver as the 25 per cent. solution, namely, 0.030 gm. Solutions of intermediate strength gave figures somewhat higher or lower. This shows that the dissolving energy of the solution is not increased by concentration. In repeating this experiment at a temperature of 50° C., or 122° F., the solubility of the silver was materially increased. The 20 per cent., 15 per cent., and 5 per cent. solutions dissolved 0.100 gm. silver each, and the others somewhat less. (Russell.)

§ 204. *Solubility of Metallic Gold.*

0.050 gm. gold-leaf were treated for forty-eight hours with 1000 c. c. of solutions of different concentration, as in § 203. In

every instance about 0.002 gm. gold were dissolved, the deviations from this quantity being exceedingly slight. No differing results were obtained with cement-gold. (Russell.)

§ 205. *The Solubility of Silver and Gold in Sodium Hyposulphite Solutions Explained.*

We have to seek the reason why concentration of the solutions does not increase the solubility of the precious metals. In order to form hyposulphite double-salts, the precious metals have to become oxidized, the oxygen being derived from the air absorbed by the solution. Hence, it is the quantity of free oxygen in the solution by which the result is, to a great extent, determined. Gold, being more difficult to oxidize than silver, is affected in a less degree. That elevation of temperature should materially favor the oxidation of the silver, is not surprising. The carbonic acid, always present in the atmosphere, may assist the reaction. For one equivalent silver dissolved, one equivalent caustic soda is set free. This is demonstrated by the fact that a neutral solution becomes caustic as soon as silver has been taken up. The quantity of silver dissolved depends upon many circumstances, but is, principally, a function of time. Leaving 200 c. c. of a 2 per cent. sodium hyposulphite solution in contact with 1 gm. cement-silver for 3 days, at a temperature of about 20° C., or 68° F., I found 0.023 gm. silver dissolved. The same experiment carried out in a stoppered flask, excluding access of air, gave only a trace of silver in solution.

§ 206. *Solubility of Metallic Copper.*

The solution of copper is effected in the same way as that of silver, both cuprous hydroxide and cuprous carbonate being soluble in sodium hyposulphite. The quantity of copper dissolved is, principally, a function of time and temperature.

200 c. c. of a solution containing 2 per cent. $Na_2S_2O_3+5aq.$, left in contact with cement-copper, dissolved :

At about 15° C., or	59° F.,	in 3 days,	0.079 gm. Cu.
" 20° C., or	68° F.,	in 24 hours,	0.030 "
" 50° C., or	122° F.,	in 2 hours,	0.011 "

If a sodium hyposulphite solution is heated to boiling in contact with copper, it dissolves much less of that metal than at a moderately warm temperature, and a precipitate of Cu_2S appears after some time. The reaction will be explained in § 308.

§ 207. *Solubility of Metallic Lead.*

A sodium hyposulphite solution, exposed to contact with the atmosphere, does not dissolve metallic lead. This is based upon the fact that both lead hydroxide and carbonate are insoluble in hyposulphite salts.

§ 208. *Solubility of Silver Oxide.*

The affinity of silver to form hyposulphite double-salts is so great that Ag_2O dissolves in a sodium hyposulphite solution, setting NaHO free. Cold solutions act with less energy than warm ones. Silver is dissolved in proportion to the quantity of sodium hyposulphite present. The solubility-coefficient, under most favorable circumstances, is about 0.200 Ag, per unit of $Na_2S_2O_3+5aq.$ (Russell.) According to the formula:

$$3Na_2S_2O_3 + Ag_2O + H_2O = 2Na_2S_2O_3,\ Ag_2S_2O_3 + 2NaHO,$$

at least a solubility-coefficient of 0.289 should be expected. Evidently, the presence of free NaHO either prevents a complete reaction, according to the above formula, or a caustic solution is a poorer solvent for the argentic double-salts than water. The deleterious influence of caustics on the solubility of silver in hyposulphite solutions is observed throughout, with the exception of a few cases only, as will be demonstrated later.

§ 209. *Solubility of Silver Sulphide.*

Silver was not dissolved after exposing 2 gm. Ag_2S to sodium hyposulphite solutions of different concentration for twenty-four hours.

§ 210. *Solubility of Gold Sulphide.*

Gold sulphide is decomposed by a solution of sodium hyposulphite. For energetic action, heating to 50° or 60° C., or to 122° or 140° F., is necessary. In treating 2 gm. Au_2S_3 for twenty-four

hours cold, 0.066 gm. gold were dissolved; while at a temperature of 65° C., or 149° F., 0.117 gm. gold went into solution in two hours. An alkaline sulphide again precipitates Au_2S_3 from this solution. (Russell.)

In explanation of these results we have to consider that gold sulphide is easily decomposed into metallic gold and sulphur. According to Levol, sulphuretted hydrogen precipitates from a boiling solution of gold chloride, metallic gold, not Au_2S_3. The gold sulphide treated in the first experiment, most likely, contained metallic gold; and in the second experiment, more metallic gold was formed. This, being in a state of exceedingly fine division, was much easier oxidized and dissolved than gold-leaf or cement-gold, used in the experiment of § 204. That Au_2S_3, as such, is not any more affected by a sodium hyposulphite solution than Ag_2S, follows from the fact that gold is again completely precipitated by alkaline sulphides.

§ 211. *Solubility of Silver Arsenate.*

Mr. Russell's results on the solubility of Ag_3AsO_4 are rather irregular, but sufficiently clear to base important conclusions upon them for practical purposes. Moderately warm solutions are much better solvents than cold ones. The solubility-coefficient found for Ag_3AsO_4, was, under most favorable circumstances, about 0.200 Ag per unit of $Na_2S_2O_3+5$ aq. The presence of caustic soda in a sodium hyposulphite solution materially increases the solubility-coefficient.

§ 212. *Solubility of Silver Antimonate.*

What has been said above, about irregularity of results, holds good in the case of $AgSbO_3$. Moderately warm solutions are decidedly better solvents than cold ones. The solubility-coefficient for warm solutions is about 0.050 Ag. per unit of $Na_2S_2O_3+5$aq. The presence of caustic soda increases the solubility of $AgSbO_3$. The effect, however, is by no means so marked as in the case of silver arsenate. The much inferior solubility of silver antimonate, as compared with the arsenate, is explained by the fact that sodium arsenate is much more soluble than the antimonate, these salts being formed in the reaction.

§ 213. *Practical Importance of the Solubility of Silver Arsenate and Antimonate in Sodium Hyposulphite Solutions.*

I pause to point out the practical importance of the facts recorded in the last two paragraphs. The majority of silver-ores, that are subjected to a chloridizing-roasting, contain antimony, or arsenic, or both. Although in chloridizing-roasting antimony and arsenic are mostly changed to chlorides, and pass off with the roasting gases, the formation of silver antimonate and arsenate can not be entirely prevented, these salts not being completely decomposed in contact with sodium chloride in the roasted ore. In treating the ore by lixiviation, the presence of these silver salts is no detriment, since they are amenable to extraction by hyposulphite solutions. In amalgamation, on the contrary, they remain mostly undecomposed, and this is the principal reason why hyposulphite solutions extract silver from tailings resulting from amalgamation of roasted ores. I have observed that the percentage of silver extracted by lixiviation from such tailings is greatly lessened if the roasting is carried on at a very high temperature. This agrees with Plattner's observation, that in fusing silver arsenate or antimonate with sodium chloride, silver chloride and the respective sodium-salts are formed.

§ 214. *Solubility of Silver Chloride.*

In dissolving AgCl by sodium hyposulphite, both double-salts mentioned at the beginning of this chapter are formed.

Two methods can be used in determining the solvent energy of sodium hyposulphite for silver chloride.

1st. By exposing freshly precipitated AgCl to the solution.

2d. By adding $AgNO_3$ to a hyposulphite solution which contains sodium chloride. In this case the AgCl is dissolved "in statu nascendi," and as soon as saturation has taken place the solution becomes turbid.

The first method gives somewhat varying results if a large surplus of AgCl is used, and left exposed to the solution for a long time. Then, the less soluble double-salt $Ag_2S_2O_3+Na_2S_2O_3+2aq.$ is formed. By using solutions of more than 10 per cent. concentration, the AgCl hardens into lumps, because in such

cases the quantity of water present is not sufficient to dissolve the argentic double-salts. For solutions of lower concentration, the solvent energy is almost entirely a function of concentration, and increases in proportion to the quantity of sodium hyposulphite present. Temperature does not influence results materially.

Mr. Russell's determinations, by the first method, give an average coefficient of 0.301 Ag, or 0.400 AgCl, for one part of $Na_2S_2O_3+5aq$.

By the second method, the coefficient is 0.365 Ag, or 0.485 AgCl, for one part $Na_2S_2O_3+5aq$. The latter figures are to be considered as the more correct ones.

It is of considerable interest to know to what degree the solubility of AgCl is influenced, if other soluble salts are mixed with the solution, or if the latter has been saturated with salts not very soluble in water, but soluble in sodium hyposulphite. The principal combinations which in this respect play an important part in the lixiviation-process, are the sulphates of lead, calcium and sodium, the caustic alkalies and alkaline earths; and these alone we shall consider.

§ 215. *Influence of Lead Sulphate on the Solubility of Silver Chloride.*

If lead sulphate has been dissolved in a sodium hyposulphite solution, and the latter is tested for its solvent energy for silver chloride, by the first method, the results show a marked decrease. It is not astonishing that this should be so, because a part of the sodium hyposulphite is consumed in the formation of the lead-salt. Assuming that this has the formula $2Na_2S_2O_3$, PbS_2O_3, and calculating the amount of sodium hyposulphite necessary in its formation, we find that the dissolving energy of the solution for silver chloride has been less reduced than should be expected, taking Mr. Russell's value of 0.400 for the solubility of silver chloride as a basis for calculation. From Mr. Russell's experiments I have deduced the following table:

Solution contains for one part $Na_2S_2O_3+5aq.$ dissolved parts of $PbSO_4$.	Decrease of dissolving energy for AgCl.	Decrease expected by the formation of $2Na_2S_2O_3$, PbS_2O_3.
0.045 parts.	8.6 per cent.	11.0 per cent.
0.054 "	11.2 "	13.2 "
0.059 "	12.6 "	14.4 "
0.086 "	15.4 "	21.0 "
0.107 "	23.8 "	26.2 "

§ 216. *Influence of Calcium Sulphate on the Solubility of Silver Chloride.*

In testing the solubility of silver chloride in a sodium hyposulphite solution saturated with gypsum, no marked decrease of solvent energy was clearly established. The reason is obvious. A double-salt of sodium and calcium hyposulphite is formed, and sodium sulphate. Any decrease in solvent energy would, most likely, be produced by the presence of the last-named salt.

The same results were obtained in saturating a calcium hyposulphite solution with gypsum.

§ 217. *Influence of Sodium Sulphate on the Solubility of Silver Chloride.*

The presence of this salt depresses the solvent energy of sodium hyposulphite for silver chloride. In experimenting with solutions containing $1\frac{1}{2}$ per cent. of $Na_2S_2O_3+5aq.$, their solvent energy was diminished 6 per cent. by addition of 2 per cent. of sodium sulphate, and 8 per cent. by addition of 5 per cent. of this salt.

§ 218. *The Effect of Caustic Alkalies and Alkaline Earths upon the Solubility of Silver Compounds.*

Mr. Russell found that the presence of a very slight quantity of a caustic alkali or an alkaline earth in a hyposulphite solution has a very deleterious effect on the extraction of silver chloride and other silver combinations from ores. This is not the case with carbonates of the alkalies. The presence of one-tenth of one per cent. of caustic soda is sufficient to depress the results in working on a large scale as much as 30 per cent. The same effect is produced by caustic lime. In adding one-half of one per cent.

of caustic lime to a sodium hyposulphite solution, the percentage of silver extracted from Ontario ore was depressed from 11 to 24 per cent. These facts seem to have been overlooked so far.

A sample of roasted Ontario ore, which yielded by lixiviation-tests with ordinary solution 86·5 per cent. of its silver, was first leached with water, and then treated for thirty-six hours with sodium hyposulphite solutions of 2½ per cent. concentration, to which various amounts of caustic soda had been added.

Per cent. of caustic soda added.	Per cent. of the silver extracted.
0.2	6.5
0.5	3.7
1.0	4.8
5.0	5.9

We note here that, after the maximum effect has been produced, an increase of caustic soda is actually beneficial. This result is produced by the increased solubility of silver arsenate and antimoniate in a caustic solution. A sodium hyposulphite solution of 2½ per cent. concentration, saturated with caustic lime, extracted from the same ore 60 per cent. of its silver. In case lixiviation-tests are made without first leaching with water, the effect of caustics is materially lessened by their action upon sulphates and chlorides of copper, zinc, and manganese.

From numerous experiments carried out by Mr. Russell, it appears that the effect of caustic soda is not so pronounced with all ores as in the example given above. More concentrated hyposulphite solutions lessen the effect; hence, lixiviation-tests in the laboratory will extract more silver from the tailings resulting from such defective lixiviation. A lixiviation-solution may become caustic:

1st. If roasted ores containing caustic lime are treated.

2d. In consequence of the presence of free caustic in the alkaline sulphide used for precipitation of the silver.

3d. If lead is precipitated as basic hydroxide by caustic lime, and a surplus of the reagent has been added.

4th. If the ore contains metallic silver or silver oxide, the solution of which in sodium hyposulphite sets caustic soda free.

The solvent energy of a caustic lixiviation-solution is restored by neutralizing it with sulphuric acid. The same effect is produced by the addition of copper sulphate.

§ 219. *Neutralizing a Caustic Stock-Solution with Sodium Bi-Carbonate.*

Mr. R. D. Clark, Superintendent of the Cortez Mill, Nevada, found accidentally that a sodium hyposulphite solution, which did not work properly, was restored to activity by addition of sodium bi-carbonate, $NaHCO_3$; and he has continued the use of the latter reagent for this purpose. In this case, a caustic lixiviation-solution is neutralized by converting caustic soda into sodium carbonate, which salt does not exert a deleterious influence on the solubility of silver compounds. The following figures, however, will show that the use of sodium bi-carbonate is not economical. To neutralize 100 pounds of caustic soda, 122½ pounds of concentrated sulphuric acid of 66° B., or 210 pounds of sodium bi-carbonate are needed. The price of sulphuric acid, including the iron tank in which it is shipped, is, in New York, at present, 1½ cents per pound, and that of sodium bi-carbonate, $3\frac{4}{10}$ cents per pound. Hence, not considering the question of freight, sodium bi-carbonate is $3\frac{2}{10}$ times dearer than sulphuric acid as a neutralizer for caustic soda.

Besides, it is not practicable to perceive, in using sodium bi-carbonate, whether the caustic soda is neutralized or not, since the reagent itself has an alkaline reaction. Hence, the bi-carbonate has to be added at random.

If the lixiviation-solution contains calcium, derived either from caustic lime or calcium sulphate in the ore, or from using calcium sulphide as precipitant, it becomes possible to ascertain the end of the reaction. From a sufficiently diluted solution, sodium bi-carbonate does not precipitate calcium, the bi-carbonate of the latter element being soluble in water. If, however, caustic soda is present, which converts the sodium bi-carbonate into the neutral salt, calcium carbonate will be precipitated so long as caustic soda remains to be neutralized.

§ 220. *The Effect of a Caustic Stock-Solution Explained.*

The deleterious effect of caustic alkalies and alkaline earths in amalgamation is well known, and forms one of several parallels we meet in both processes.

In explanation of the reactions interfering with the solution of silver in lixiviation, I suggest the following: Caustics seem to weaken the affinity of silver to form hyposulphite double-salts, as already pointed out.

Then, the silver chloride in roasted ore is evidently more or less fused together with lead chloride and lead sulphate, which are only imperfectly removed by leaching with water. If then a caustic hyposulphite solution is brought in contact with the ore, the caustic soda reacts upon the base-metal salts, converting them into hydroxides. The latter form an insoluble envelope round the silver chloride, preventing its complete solution. It does not matter whether the lixiviation-solution is originally caustic, or becomes so in contact with ore containing caustic lime.

Hence, we would have both a chemical and a mechanical cause explaining the phenomenon.

§. 221. *Solubility of the Carbonates of Silver, Copper, Lead, and other Metals.*

Lead carbonate is insoluble in a sodium hyposulphite solution, while silver carbonate and cuprous carbonate are soluble. Hence, if sodium carbonate is added to a hyposulphite solution containing lead, silver, and copper, $PbCO_3$ alone is precipitated. If a solution of $PbSO_4$ in sodium hyposulphite is so treated, the precipitation of the lead is so complete, that H_2S gives no reaction in the filtrate. Cupric carbonate is also insoluble, but since cupric hyposulphites do not exist, a precipitation of the cupric salt can not take place. Upon these reactions is based the separation of lead from silver and copper in Russell's lixiviation-process. In effecting this on a large scale, it is most economical to use the pure soda manufactured by the Solvay or ammonia process. If ordinary soda-ash is used, which contains more or less Na_2S, the solution must be purified, otherwise Ag_2S would be precipitated with $PbCO_3$. This is effected by adding to the soda-ash solution the so-called extra-solution, (see next chapter) until a precipitate of copper sulphide ceases to appear. An excess of the reagent is not injurious, and it is not at all necessary, although better, to dissolve the soda-ash in a sodium hyposulphite solution. This is a much better method of purifying soda-ash than the one described in my first paper on

the Russell process. If the soda-ash is dissolved in a hyposulphite solution, and then copper sulphate is added, cupric carbonate, and also cupric hydroxide, are precipitated besides copper sulphide. The cupric hydroxide is derived from caustic soda generally contained in soda-ash. The presence of caustic soda is not injurious since it also precipitates lead. In the method first described, neither cupric carbonate, nor hydroxide, but only copper sulphide is precipitated. If the soda-ash has not been dissolved in a sodium hyposulphite solution, the extra-solution used for precipitating the sulphur should contain for one part $CuSO_4+5aq.$, at least $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$

The carbonates of iron, manganese, zinc, and calcium are also insoluble in a sodium hyposulphite solution. In well roasted ores only traces of iron salts exist, and these, as well as the chlorides and sulphates of zinc and manganese, are removed by the wash-water. Hence, none of these carbonates are precipitated with the lead carbonate.

If lead is to be precipitated by this process, the use of calcium hyposulphite is not admissible, neither can calcium sulphide be used for precipitation of the silver.

Should roasted ore contain calcium sulphate and caustic lime in such quantities that their complete removal by the first wash-water is impracticable, then the lixiviation-solution will hold calcium salts, and calcium carbonate will be precipitated together with the lead in attempting to separate this metal.

If a sodium hyposulphite solution is impregnated with a sufficient quantity of sodium carbonate, prior to its application to lead-bearing ores, lead does not go into solution, but remains as a carbonate in the charge with the tailings. Wherever lead carbonate has a commercial value, this method is not economical. Its only advantage would be that a separate precipitation of the lead is avoided. That the presence of sodium carbonate also prevents calcium from entering the lixiviation-solution, is hardly necessary to point out. I have already mentioned that sodium bi-carbonate, $HNaCO_3$, is used in the Cortez Mill for neutralizing a lixiviation-solution containing caustic soda. The secondary effect of this reagent is that it prevents the extraction of lead from the ore, sodium bi-carbonate precipitating lead as well as sodium carbonate. Here again the dearer reagent is used in preference to the cheaper

one. 100 pounds Na_2CO_3 are equivalent in effect to 158 pounds $HNaCO_3$, since two equivalents of the latter salt are required to precipitate one equivalent of lead from hyposulphite solutions, an acid sodium hyposulphite not being in existence. The price of Solvay soda is $1\frac{4}{10}$ cents; that of sodium bi-carbonate, $3\frac{4}{10}$ cents per pound. This makes the latter salt $3\frac{8}{10}$ times dearer than the former as a precipitant for lead, not taking into account the difference in freight.

In experimenting with this process, Mr. Russell found that if sodium carbonate is added at once to the lixiviation-solution to prevent the extraction of lead, a much larger quantity of the reagent is consumed than in precipitating the lead after lixiviation is completed. This is explained by the fact that a diluted sodium hyposulphite solution will dissolve only a limited amount of lead sulphate. If, at the same time, sodium carbonate is present, the dissolving and precipitating actions of sodium hyposulphite and sodium carbonate go on indefinitely until the larger portion, or all of the lead sulphate in the ore, has been converted into carbonate. The same statement holds good in regard to calcium. In treating ores containing a considerable quantity of calcium sulphate and caustic lime, the separation of lead by this process ceases to be economical. This will become evident from the following figures. The precipitation of one pound lead required 0.512 pounds Na_2CO_3, while for one pound calcium, 2.65 pounds Na_2CO_3 have to be consumed according to chemical equivalents.

That sodium bi-carbonate does not precipitate calcium from a lixiviation-solution, unless the latter is caustic, has already been pointed out in § 219. Hence, in using this reagent at the Cortez Mill, caustic lime is prevented from entering the lixiviation-solution, but not calcium sulphate.

§ 222. *Solubility of Lead Hydroxide and Cuprous Hydroxide.*

Hydroxide of lead is insoluble, while cuprous hydroxide and silver oxide are soluble in hyposulphite solutions. Upon this is based the separation of lead from copper and silver by caustic lime. To prevent precipitation of copper, the solution should

contain for one part metallic copper not less than eighteen parts sodium hyposulphite. The separation of lead from silver by caustic lime is very incomplete. According to Mr. C. Rueger, the lead precipitate, resulting at the Mount Cory Mill, Nevada, contained 60 per cent. lead, 420 ounces, and more, silver per ton, besides a considerable amount of gypsum, impurities of the lime, and excess of caustic lime. Although I have failed to obtain detailed information about the working of this process, it appears to me that it compares very unfavorably with the separation of lead by sodium carbonate.

The solubility of caustic lime in water, and also in hyposulphite solutions, is so slight that the reagent must be used in the form of milk of lime, made with a hyposulphite solution in preference to water. If a surplus of the reagent is added, lead will again go into solution, lead hydroxide being soluble in caustics.

Where Na_2S_2 is used as precipitant for sulphides, and the ore itself is free from calcium-salts, the adoption of this process would be an absurdity. Even if calcium compounds are present in the lixiviation-solution, and sodium carbonate can not be used with economy for the separation of lead, it is very doubtful whether it is profitable to precipitate lead by caustic lime.

§ 223. *Solubility of the Oxides and Hydroxides of Copper, Iron, Zinc, and Manganese.*

Cupric oxide and hydroxide, the oxides and hydroxides of iron, manganese, and zinc, are all insoluble in hyposulphite solutions.

§ 224. *Solubility of Cuprous Chloride.*

Cuprous chloride, Cu_2Cl_2, is dissolved by sodium hyposulphite with the formation of double-salts, the nature of which will be fully considered in the next chapter.

§ 225. *Solubility of Lead Sulphate.*

Lead forms a sodium hyposulphite double-salt of the formula $2Na_2S_2O_3$, PbS_2O_3, without water. The solubility of lead sulphate in sodium hyposulphite is characteristic, on account of its being

principally a function of the concentration of the solution. If the same quantity of sodium hyposulphite is dissolved in varying quantities of water, the solubility of the lead sulphate increases materially with the concentration of the solution. The following figures, deduced from a series of Mr. Russell's experiments, will illustrate this.

Concentration of the solution in $Na_2S_2O_3+5aq.$	Amount of $PbSO_4$ dissolved for one part of $Na_2S_2O_3+5aq.$
5 per cent.	0.080 parts.
6.6 "	0.102 "
10 "	0.131 "
20 "	0.194 "
25 "	0.226 "

This peculiarity is very favorable for working on a large scale, where hyposulphite solutions of low concentration are used, and the quantity of lead sulphate dissolved is thus brought to a minimum.

The solubility of lead sulphate increases materially with the temperature of the solution. In exposing 20 gm. $PbSO_4$ to 1000 c.c. of a 5 per cent. sodium hyposulphite solution for one hour, 0·048 parts of $PbSO_4$ per unit of $Na_2S_2O_3+5aq.$ were dissolved at 21° C., or 70° F., while the solubility-coefficient at 49° C., or 120° F. was 0.095.

§ 226. *Solubility of Lead Antimonate.*

The solubility of lead antimonate is slight.

1000 c.c. of a 2 per cent. sodium hyposulphite solution dissolved:

At 20° C., or 68° F., 0.147 gm. $PbSb_2O_6$.
" 75° C., or 167° F., 0.306 gm. "

§ 227. *Solubility of Calcium Sulphate.*

If calcium sulphate is dissolved in a sodium hyposulphite solution, sodium sulphate, and a double-salt of sodium and calcium hyposulphite, are formed.

The solubility of calcium sulphate or gypsum per unit of sodium hyposulphite, is principally a function of the concentration of the solution. The more dilute solutions are relatively better solvents than more concentrated ones. The following figures are

deduced from a series of Mr. Russell's experiments. The solubility of calcium sulphate is decreased by elevating the temperature of the sodium hyposulphite solution, as is the case in dissolving gypsum in water. [One part of $CaSO_4+2aq.$ requires 400 parts of cold water, and 460 parts of boiling water for its solution.]

TABLE No. 227.

SHOWING SOLUBILITY OF CALCIUM SULPHATE IN SODIUM HYPOSULPHITE SOLUTIONS.

Concentration of the Solution in $Na_2S_2O_3+5aq.$	One part $CaSO_4+2aq.$ required solution :	Amount of $CaSO_4+2aq.$ dissolved for one part $Na_2S_2O_3+5aq.$
5 per cent.	101 parts.	0.198 parts.
6.6 "	84.8 "	0.181 "
10 "	66.2 "	0.151 "
20 "	33.8 "	0.148 "

The experiments were made at about 18° C., or 64.4° F. (Russell).

§ 228. *Solubility of Caustic Lime.*

Caustic lime dissolves in sodium hyposulphite solution, a double-salt of sodium and calcium hyposulphite, and free caustic soda, being formed. Its solubility, however, is not much greater than that in water, diminishing, as in the latter case, with increase of temperature.

The following quantities of CaO were dissolved by 1000 c. c. of a solution containing 2 per cent. $Na_2S_2O_3+5aq.$:

At 22° C., or 71.6° F., 1.975 gm. CaO.
" 75° C., or 167° F., 1.295 " "
" 100° C., or 212° F., 1.160 " "

I have not tested solutions of higher concentration in sodium hyposulphite, because they are not used in practice.

§ 229. *Solubility of Sodium Antimonate.*

Potassium antimonate, added to a solution of sodium hyposulphite, produces the same characteristic precipitate of $Na_2Sb_2O_6 + 7aq.$ that is obtained with other sodium-salts. The precipitate appears even in very diluted solutions; hence, it must be assumed that the solubility of this salt in sodium hyposulphite is not greater than in water.

1000 c. c. boiling water dissolve 2.86 gm. $Na_2Sb_2O_6 + 7aq.$

Cold water dissolves less.

CHAPTER III.

The Reactions of Extra-Solutions.

§ 301. *Cuprous Hyposulphite Double-Salts.*

If solutions of alkaline hyposulphites are added to salts of cupric oxide, hyposulphite double-salts, containing the alkali and cuprous oxide, are formed. If the solutions are not too dilute, these double-salts separate in the form of canary-yellow precipitates.

The latter, under certain conditions, turn black in consequence of the formation of copper sulphide. By treating them with boiling water, the cuprous hyposulphite is completely decomposed to sulphide. Dilute acids do not decompose these double-salts at ordinary temperature, but do so very quickly upon heating. Some of these double-salts are very little soluble in water, but dissolve readily in aqueous hyposulphite, forming new salts with higher equivalents of the alkaline hyposulphites.

The latter solutions can be heated to 85° C., or 185° F., before Cu_2S commences to separate.

The potassium-salt has, according to Rammelsberg, the formula $K_2S_2O_3$, $Cu_2S_2O_3+2aq$. Lenz obtained a sodium double-salt of the formula $2Na_2S_2O_3$, $3Cu_2S_2O_3+5aq$. Its formation is expressed by the equation

(No. 1.) $$11Na_2S_2O_3+6CuSO_4=2Na_2S_2O_3,\ 3Cu_2S_2O_3+6Na_2SO_4+3Na_2S_4O_6,$$

leaving out the water-equivalents. It is a reaction similar to the one which takes place if sodium hyposulphite is treated with iodine, as far as the formation of sodium tetrathionate is concerned, viz:

$$2Na_2S_2O_3+2I=2NaI+Na_2S_4O_6.$$

§ 302. *The Standard Extra-Solution.*

The salt $2Na_2S_2O_3$, $3Cu_2S_2O_3+5aq$. is very little soluble in water; 1 part requires 352 parts of water for solution at ordinary temperature. In aqueous sodium hyposulphite, it dissolves freely, as the figures below demonstrate:

100 c. c. of a 5 per cent. solution dissolve 12.28 gm.
" " $7\frac{1}{2}$ " " " 17.46 "
" " 10 " " " 22.54 "

This gives 2.46 gm., 2.32 gm., and 2.25 gm., of $2Na_2S_2O_3$, $3Cu_2S_2O_3+5aq.$ per gm. of $Na_2S_2O_3+5aq.$ respectively.

In calculating the atomic weights, I find that it takes nearly 2 equivalents of sodium hyposulphite to dissolve 1 equivalent of the $\frac{2}{3}$ salt of Lenz. Hence, a double-salt of the formula $4Na_2S_2O_3$, $3Cu_2S_2O_3+x$ aq. must be assumed to exist in the solution.

The same salt would be formed by satisfying the equation:

(No. 2.) $13Na_2S_2O_3+6CuSO_4=4Na_2S_2O_3, 3Cu_2S_2O_3+6Na_2SO_4+3Na_2S_4O_6,$

namely, by mixing solutions containing for 1 part $CuSO_4+5aq.$, $2\frac{15}{100}$ parts $Na_2S_2O_3+5aq.$ A solution of this composition is called a *Standard Extra-Solution.* In actual lixiviation the proportions of the chemicals for this solution are generally so regulated that for each $2\frac{1}{4}$ parts $Na_2S_2O_3+5aq.$, 1 part $CuSO_4+5aq.$ is added. In using these proportions of the reagents, precipitation of cuprous hyposulphite is prevented.

If the salt of Lenz is dissolved in an excess of sodium hyposulphite, alcohol precipitates from the solution a double-salt of the formula:

$$3Na_2S_2O_3,\ Cu_2S_2O_3+2aq.$$

All the following reactions, which I have carefully investigated, refer to extra-solutions of moderate concentration, as used in actual lixiviation on a large scale. Extra-solutions of standard composition, and those containing a much larger quantity of sodium hyposulphite in proportion to copper sulphate, differ so materially in their properties, that I shall consider them separately.

I. Reactions of Extra-Solutions with Caustic Soda and Sodium Carbonate.

A. Extra-Solution of Standard Composition, containing for 1 part $CuSo_4+5aq.$, $2\frac{1}{4}$ parts $Na_2S_2O_3+5aq.$

§ 303. *Reactions with Caustic Soda.*

Caustic soda precipitates, at ordinary temperature, cuprous hydroxide, CuHO. The precipitate appears, after some time,

even in very diluted solutions. The separation of copper is, however, by no means complete. Upon addition of a sufficient amount of sodium hyposulphite, the precipitate is again dissolved. If the cuprous hydroxide is exposed to the air, it is converted into cupric hydroxide, $Cu(HO)_2$, which is insoluble in sodium hyposulphite.

§ 304. *Reactions with Sodium Carbonate.*

Sodium carbonate does not produce, at ordinary temperature, a precipitate. Upon heating the solution above 30° C., or 86° F., a precipitate of cuprous carbonate, Cu_2CO_3, appears, having the same color as the cuprous hydroxide. It is again dissolved upon addition of sodium hyposulphite. According to Gmelin, cuprous hydroxide, and not carbonate, is precipitated from cuprous salts by alkaline carbonates.

If to a sodium hyposulphite solution either caustic soda or sodium carbonate are added, and then copper sulphate, the characteristic blue precipitate of either cupric hydroxide, $Cu(HO)_2$, or of basic cupric carbonate, $CuCO_3$, $Cu(HO)_2$, appear, both entirely insoluble in sodium hyposulphite.

Extra-solution containing at least twice the quantity of sodium hyposulphite required for standard composition, does not produce a precipitate in a solution of sodium carbonate.

B. The Extra-Solution contains for 1 part $CuSO_4+5aq.$, $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$

§ 305. *Reactions with Caustic Soda.*

Caustic soda does not produce a precipitate at ordinary temperature. The latter appears, however, upon heating the solution to about 50° C., or 122° F. If the extra-solution contains more than $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$ for 1 part $CuSO_4+5aq.$, a higher temperature is required to separate cuprous hydroxide or oxide, but the reaction always takes place before the boiling-point has been reached.

§ 306. *Reactions with Sodium Carbonate.*

Sodium carbonate does not produce a precipitate of cuprous carbonate, even if the solution is heated to boiling.

II. Decomposition of Extra-Solutions by Heating.

§ 307. *A. Extra-Solutions of Standard Composition.*

An extra-solution without free acid commences to decompose, if heated above 85° C., or 185° F. A higher temperature rapidly increases decomposition, and upon boiling, the copper separates completely as Cu_2S, free sulphuric acid being formed under evolution of sulphurous acid. Solutions with a slight amount of free acid commence to turn dark at 65° C., or 149° F.; and the more free acid is present, the lower is the temperature at which decomposition begins.

§ 308. *B. Extra-Solutions containing for* 1 *part* $CuSO_4 + 5aq.$, $4\frac{1}{2}$ *parts* $Na_2S_2O_3 + 5aq.$

Extra-solutions of this composition, if free from acid, can be heated to boiling before they turn dark. Even then decomposition is very slow. I have stated in § 206, that in heating ordinary sodium hyposulphite solution with metallic copper, cuprous hyposulphite remains in solution even after continued boiling. This is explained by the facts stated above.

If the extra-solution contains free sulphuric acid, Cu_2S is separated at a much lower temperature, the degree of the latter depending upon the quantity of free acid present. In all cases the energy of the reaction, after it has once commenced, is increased by the formation of more free acid.

III. Decomposition of Extra-Solutions at Ordinary Temperature.

Freshly made extra-solutions always show an acid reaction, even if the reagents used were perfectly pure. The commercial bluestone generally encloses acid mother-solution; hence, extra-solution, prepared from it, may contain free acid. Such solutions can be completely neutralized by sodium carbonate or caustic soda.

It is a great mistake to assume that an extra-solution necessarily should show an acid reaction. It can be made alkaline by sodium carbonate, or, under certain conditions, caustic by caustic soda.

All extra-solutions undergo decomposition in contact with the atmosphere on account of the great affinity of cuprous salts for oxygen.

A. Extra-Solutions of Standard Composition.

§ 309. *Solutions without free Acid.*

Solutions without free acid become turbid immediately after mixing. Very soon, a precipitate of cupric hydroxide appears, which becomes very copious after 24 hours. The reaction can be explained by the following formulas:

$$2Cu_2S_2O_3 + 3O + 3H_2O = CuS_4O_6 + 3Cu(HO)_2.$$

The cupric tetrathionate, "in statu nascendi," decomposes sodium hyposulphite:

$$2CuS_4O_6 + 3Na_2S_2O_3 = Cu_2S_2O_3 + 3Na_2S_4O_6.$$

These reactions go on until, finally, nearly all the copper is precipitated at hydroxide, and most of the sodium hyposulphite is converted into tetrathionate:

$$Cu_2S_2O_3 + Na_2S_2O_3 + 2O + 2H_2O = 2Cu(HO)_2 + Na_2S_4O_6.$$

§ 310. *Acid Solutions.*

If a few drops of diluted sulphuric acid are added to the solution, so as to give it a decided acid reaction, it becomes perfectly clear, and remains so for over 24 hours. It would be a mistake to assume that decomposition does not go on. The same reactions take place as just pointed out above, with this difference, that deterioration in cuprous hyposulphite is prevented so long as free acid is present. The oxygen absorbed by the cuprous salt is all transferred to the sodium hyposulphite. After sufficiently long exposure to the atmosphere, the free acid becomes neutralized by $Cu(HO)_2$, and the same reactions now go on as in a neutral solution. By this time, the solution has so deteriorated in sodium hyposulphite, that a precipitate of the canary yellow $\frac{2}{3}$ salt of Lenz sometimes appears, besides cupric hydroxide. The decomposition of an acid solution is slow as compared with a neutral one. It may be considered as quite permanent for practical pur-

poses, at least so far as cuprous hyposulphite is concerned. A solution with too much free acid is liable to undergo the same alteration, at ordinary temperature, as described in § 307, with separation of Cu_2S.

§ 311. *Solutions to which Sodium Carbonate has been added.*

If sodium carbonate is added to the solution, decomposition is so rapid that the filtrate, under access of air, becomes again turbid as soon as it leaves the funnel. Precipitation of cupric carbonate continues until the sodium carbonate is exhausted. In this case the solution suffers only deterioration in cuprous hyposulphite, but not in sodium hyposulphite. The effect is the same as if a cupric salt is added to a hyposulphite solution containing sodium carbonate. Cupric carbonate is formed before a reaction between the cupric and hyposulphite salts can take place. In adding sodium carbonate to a standard extra-solution, a precipitate of cupric carbonate appears so soon, that a careless observer might be led to assume that the solution contained a cupric salt, which is not possible.

B. Extra-Solutions containing for 1 part $CuSO_4+5aq.$, $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$

These solutions are clear and colorless after mixing.

§ 312. *Solutions without free Acid.*

Solutions without free acid remain clear for many days, without depositing cupric hydroxide. That a deterioration in sodium hyposulphite does take place, after several days' exposure, becomes evident upon heating the solution, when it decomposes rapidly, as described in § 307. A freshly made solution, it will be remembered, decomposes very slowly upon boiling. Finally, these solutions deposit Cu_2S, and the more concentrated they are the sooner commences this reaction.

§ 313. *Acid Solutions.*

Solutions to which sulphuric acid has been added, soon turn yellow. If the quantity of acid is very slight, the color of the solution does not change for many days, deterioration going on in

sodium hyposulphite only. In the presence of more acid, decomposition, as described in § 307, sets in sooner or later, and, if once started, the copper is rapidly precipitated as Cu_2S.

§ 314. *Solutions to which Sodium Carbonate has been added.*

Solutions to which sodium carbonate has been added, either remain clear or become slightly turbid, but, in the latter case, remain clear after filtering. In the decomposition which these solutions undergo, their concentration has some influence. A solution of $2\frac{1}{2}$ per cent. concentration in sodium hyposulphite and $\frac{1}{2}$ per cent. in copper sulphate, does not change, apparently, after many days' exposure. Its deterioration in sodium hyposulphite is, however, proven by the fact, that it deposits cuprous oxide upon heating, which is not the case with a fresh solution.

Solutions of $4\frac{1}{2}$ per cent. concentration in sodium hyposulphite and $\frac{1}{2}$ per cent. in copper sulphate, act differently. After 24 hours, they assume the color of rich port-wine, indicating the presence of Cu_2S. Indeed, upon boiling, a trace of Cu_2S separates and the solution becomes again colorless. It is difficult to explain why this decomposition should take place in an alkaline solution, at ordinary temperature. This decomposition, however, proceeds very slowly.

Upon boiling the solution, after many days' exposure, it becomes turbid, and a more copious precipitate of Cu_2S is formed.

§ 315. *Solutions to which Caustic Soda has been added.*

Solutions to which caustic soda has been added show exactly the same phenomena as described in § 311, with this difference, that cupric hydroxide is precipitated in place of cupric carbonate. Deterioration in cuprous hyposulphite goes on until the caustic soda is exhausted, while sodium hyposulphite remains intact. In solutions with greater excess in sodium hyposulphite, the effect of the caustic soda is materially weakened.

§ 316. *The Difference in Reactions Explained.*

It may be asked—what is the reason that extra-solutions of standard composition, and those containing twice the amount of sodium hyposulphite, differ so materially in their reactions? We have

to deal with two entirely different double-salts. The one has the formula $4Na_2S_2O_3$, $3Cu_2S_2O_3$; the other, $8Na_2S_2O_3$, $3Cu_2S_2O_3$. The former is slightly yellow; the latter colorless. The latter salt is also the most permanent of the two, but for that reason it acts with less energy on silver-compounds.

§ 317. *Conclusions of Practical Importance.*

The facts elucidated above are of great practical value. They demonstrate:

1st. That a standard extra-solution is most permanent, at ordinary temperature, if it contains a slight quantity of free sulphuric acid. Such solutions can be safely heated to 50° C., or 122° F.

2d. A standard extra solution should never be prepared from an alkaline or caustic stock-solution (containing sodium carbonate or caustic soda). Not only is thereby caused a waste in copper sulphate, the copper being precipitated as cuprious hydroxide or carbonate, but the solution becomes neutral, and, in consequence, liable to speedy deterioration.

3d. An extra-solution containing twice the quantity of sodium hyposulphite, required for standard composition, is most permanent if neutral, free acid producing, at ordinary. temperature, decomposition with formation of Cu_2S. Such solutions, without free acid, are especially recommended for application at higher temperatures than 50° C., or 122° F.

4th. If ores containing caustic lime are treated, extra-solution of standard composition deteriorates so quickly that its use becomes unprofitable. Hence, the application of solutions low in copper, but high in sodium hyposulphite, is most judicious until the deleterious influence of the caustic lime has been counteracted and lixiviation with ordinary solution is commenced. This agrees with practical experience.

5th. It is evident that extra-solution cannot be kept in stock, but must be prepared shortly before application.

6th. If roasted ores, containing cuprous chloride, are lixiviated, the ordinary solution will deteriorate more quickly in hyposulphite than if cuprous chloride is absent. This is one of the reasons why the loss in sodium hyposulphite is greater in treating acid—namely, ores producing acid wash-water—than alkaline ores, cup-

rous chloride being decomposed by caustic-lime, if present in the latter case.

The investigation of the reactions of extra-solution is interesting both from a scientific and a practical standpoint, but much remains to be done to exhaust the subject.

§ 318. *Investigation on the Formation of Sulphates in the Decomposition of Extra-Solutions.*

In elucidating the decomposition of extra-solutions by atmospheric influences, the formation of tetrathionates occurs so frequently that the question may be asked, whether sulphates are not formed as well. The latter is, in my opinion, only the case where a decomposition with production of Cu_2S takes place. To prove the correctness of this argument, the following experiment was made. Extra-solution, free from sodium sulphate, was prepared by dissolving the pure $\frac{2}{3}$ salt of Lenz in sodium hyposulphite. $BaCl_2$ precipitates from a moderately concentrated solution of this composition BaS_2O_3, which, however, readily dissolves upon addition of water. The same is the case if $BaCl_2$ is added to a tetrathionate solution. In both of these solutions, containing barium, the slightest quantity of Na_2SO_4 produces a precipitate of $BaSO_4$, proving that this reaction is reliable. The extra-solution was now exposed for 48 hours to the oxidizing influence of the atmosphere, after which time not a trace of sulphate could be detected.

§ 319 *Silver Tetrathionate.*

Another subject needed investigation, namely, the solubility of silver chloride in sodium tetrathionate. To test this, concentrated solutions of copper sulphate and sodium hyposulphite were mixed in proportion to form the insoluble $\frac{2}{3}$ salt of Lenz. After separation of the latter, the filtrate, containing Na_2SO_4, $Na_2S_4O_6$, and a slight quantity of the salt of Lenz, was digested with AgCl. The latter was rapidly converted into a heavy, crystalline salt of $Ag_2S_4O_6$. This salt is of so little stability that it decomposes to Ag_2S at very moderate heat. Sodium hyposulphite dissolves it readily. The filtrate from the silver tetrathionate was made acid, and heated to boiling. A slight quantity of Cu_2S and Ag_2S separated. Their solution in nitric acid gave with hydrochloric acid

so small a precipitate of silver chloride, that its weight was not determined. This proves conclusively that sodium tetrathionate is not a solvent for silver, and that all tetrathionate formed in the preparation of extra-solution is impoverishing the solvent energy of a lixiviation-solution.

§ 320. *The Effect of Extra-Solutions upon Silver-Compounds and Silver-Minerals, Compared with Amalgamation.*

The solutions of the cuprous hyposulphite double-salts are peculiar on account of the energetic decomposing action they exert upon most silver-combinations and silver-minerals. That upon silver sulphide has been especially investigated. If the silver sulphide has been freshly presipitated, the reaction is practically instantaneous. In all cases an argentic hyposulphite double-salt is formed, which goes into solution, and copper sulphide is precipitated in case the silver was combined with sulphur.

The most energetic reactions are produced by an extra-solution of standard composition, an increase in sodium hyposulphite as well as in copper sulphate diminishing the effect.

We have here an interesting parallel with the action of cuprous chloride on silver sulphide, and various silver-minerals, upon which the process of raw amalgamation is principally based. But there is this difference, that cuprous hyposulphite acts very energetically, even in dilute solutions, while the cuprous chloride must be used rather concentrated to decompose most silver-minerals, except the sulphide, whereby it becomes too expensive for practical operations, and also produces a silver-bullion of low fineness.

§ 321. *Extra-Solution with Cuprous Chloride.*

The equations, No. 1 and No. 2, show that in forming extra-solution, one equivalent of $Na_2S_2O_3+5aq.$ is destroyed for each equivalent of $CuSO_4+5aq.$ consumed. This is not the case if cuprous chloride is used in the preparation of extra-solution.

$$7Na_2S_2O_3+3Cu_2Cl_2=4Na_2S_2O_3,\ 3Cu_2S_2O_3+6NaCl.$$

The extra expense of preparing cuprous chloride would not compensate for the saving in sodium hyposulphite, so that this method cannot be adopted in practice. The reaction, however, has

a practical bearing in the treatment of ores containing copper. In the chloridizing-roasting of such ores, more or less cuprous chloride is formed, the amount increasing, the lower the temperature at which the roasting was conducted.

Base-metal leaching with cold water leaves most of the cuprous chloride in the charge. In the subsequent lixiviation for silver, the hyposulphite solution dissolves the cuprous chloride, thus forming a weak extra-solution. Although the effect of this extra-solution cannot be compared with one of standard composition, it nevertheless has a beneficial effect upon the extraction of the silver. This is one of the reasons why cupriferous silver-ores are more amenable to ordinary lixiviation than those without copper.

§ 322. *Extra-Solution made by Dissolving the $\frac{2}{3}$ Salt of Lenz in Sodium Hyposulphite.*

According to Mr. Russell's observations, the $\frac{2}{3}$ salt of Lenz, $2Na_2S_2O_3$, $3Cu_2S_2O_3+5aq.$, is quite permanent. It can be kept moist, if excluded from the atmosphere, or dried at about 40° C., or 104° F., without decomposing. At a much higher temperature, it decomposes rapidly. For production of 100 pounds of this salt, ~~240~~ 243.9 pounds sodium hyposulphite and ~~60~~ 132.3 pounds copper sulphate are required; a total of ~~270~~ 376.2 pounds chemicals. All the copper in the copper sulphate is contained in the cuprous double-salt. In preparing the extra-solution in the usual way, the sodium sulphate and tetrathionate add simply so much inert and useless matter to the lixiviation-solution. This brings up the question, whether it would not be practicable, and very much cheaper, where freight is unusually high, to ship 100 pounds of the $\frac{2}{3}$ salt of Lenz, in place of ~~270~~ 376.2 pounds chemicals, and then make the extra-solution by dissolving the cuprous hyposulphite in sodium hyposulphite. The salt of Lenz could be obtained in the form of cakes, with about 20 per cent. of water, outside of that chemically combined, by means of a filter-press, at slight expense. Freight would be saved on about ~~160~~ 129 pounds. In a mill of 60 tons daily capacity, using 5 pounds copper sulphate per ton of ore, there would be a saving on about ~~25,000~~ 17,974 pounds freight per month. Besides this great saving, the stock-solution would not be overloaded with foreign salts.

The following figures will be useful:

1 pound of the $\frac{3}{8}$ salt of Lenz requires $\frac{1}{2}$ pound sodium hyposulphite to form standard extra-solution.

$\frac{0.75}{1.00}$ pounds of the $\frac{3}{8}$ salt of Lenz are equal in effect to 1 pound copper sulphate.

Another point, however, needs consideration. In making the extra-solution in the ordinary way, 1 pound sodium hyposulphite is destroyed for 1 pound copper sulphate consumed, while $\frac{0.75}{1.00}$ pounds of the salt of Lenz, add $\frac{0.83}{1.00}$ pounds sodium hyposulphite to the stock-solution after it passes the precipitating-tanks. Hence, if ores are lixiviated requiring a large quantity of cuprous hyposulphite for treatment by extra-solution, the use of the $\frac{3}{8}$ salt of Lenz will add more sodium hyposulphite to the solution than necessary to counterbalance its deterioration. This demonstrates how carefully in lixiviation all economical questions have to be examined.

Solubility of Silver, Gold, Lead, Silver Sulphide, and Gold Sulphide in Extra-Solution.

§ 323. *Solubility of Metallic Silver.*

Metallic silver is much easier dissolved by a standard extra-solution than by ordinary solution. As is the case with the latter, concentration of the extra-solution does not materially affect the result. Elevation of temperature again facilitates the solution of the silver. From a number of Mr. Russell's experiments it appears that the best effect, under equal conditions, obtained with extra-solution on metallic silver, is about nine times as great as with ordinary solution, if both are used cold; and about three and a half times as great if the solutions are heated to 50° C., or 122° F.

Upon what chemical reaction is the greater solubility of metallic silver in extra-solution based, as compared with ordinary solution? Evidently, copper can not be replaced by silver, because the latter is precipitated by the former from a hyposulphite solution. The copper in the extra-solution acts only a secondary part in facilitating the oxidation of the silver. All cuprous salts have a strong tendency to absorb oxygen from the atmosphere, and to

act as carriers of this element for oxidation of other substances. That a cuprous hyposulphite solution possesses this property in a high degree, has already been fully demonstrated. The reaction goes on indefinitely as long as reducing and oxidizing agents are present simultaneously. It is also evident that concentration of the solution, within certain limits, can not influence the result. The absorption of oxygen by the solution is principally a function of time, temperature, and surface exposure. Standard extra-solutions absorb oxygen much quicker than those containing an excess of sodium hyposulphite. For this reason they are the most energetic solvents for metallic silver.

§ 324. *Solubility of Metallic Gold.*

A series of experiments was made as in § 204, with substantially the same results, showing the dissolving energy of extra-solution upon metallic gold to be equal, not superior, to that of ordinary solution. (Russell.) The slight affinity of gold for oxygen, which is not sufficiently stimulated by the presence of a cuprous salt, explains this fact. A different result may be expected in treating gold not in the form of gold-leaf, or cement-gold, but in finely divided condition, as obtained by reduction from gold sulphide.

§ 325. *Solubility of Metallic Lead.*

Metallic lead precipitates copper from standard extra-solutions quite rapidly, even at ordinary temperature, especially if free acid is present. In solutions with an excess of sodium hyposulphite, the reaction is decidedly slower, and in the absence of free acid, it is only observed after some time.

326. *Solubility of Silver Sulphide.*

Ag_2S is readily and rapidly decomposed, and the silver dissolved, if treated with extra-solution, especially if the latter is warm. Standard extra-solutions were prepared by saturating sodium hyposulphite solutions of various concentration with the $\frac{2}{3}$ salt of Lenz.

The effect of such solutions on Ag_2S was as follows, namely:

Saturated solution of the ⅔ salt of Lenz in aqueous sodium hyposulphite of	Silver dissolved per gm. of the ⅔ salt of Lenz.
5 per cent. concentration.	0.137 gm.
7½ " "	0.115 "
10 " "	0.113 "

A solution of the ⅔ salt of Lenz in water dissolved only 0.022 gm. silver per gm. of this salt.

The equivalent of ⅔ salt of Lenz is 1120; that of silver, 108. If all the copper had been replaced by silver, 0.578 gm. Ag should have gone into solution per gm. of the ⅔ salt of Lenz. The actual result is less than ¼ of the above figure. It has to be considered, however, that in order to effect a complete decomposition, there is not sufficient sodium hyposulphite in a standard extra-solution to form the easily soluble double-salt, $2Na_2S_2O_3$, $Ag_2S_2O_3$. On this basis a solution of 0.385 gm. Ag might be expected. Numerous experiments made by Mr. Russell have established the fact that an increased quantity of sodium hyposulphite, in place of augmenting, materially diminishes the solvent energy of extra-solution. Hence, deficiency in sodium hyposulphite, necessary for the formation of the easily soluble argentic double-salt, can not be the cause that the quantity of silver dissolved is so small. Evidently, the reaction is checked, after a certain amount of silver has entered the solution. This is confirmed in working on a large scale. It is always necessary to use an excess of extra-solution to dissolve silver-compounds that can not be extracted by ordinary lixiviation. A more logical explanation of these observations can be given as follows. It has been demonstrated that the permanency of the cuprous hyposulphite double-salts increases, the more equivalents of sodium hyposulphite they contain in proportion to cuprous hyposulphite. The least permanent extra-solution is one of standard composition; hence, it acts most energetically as a solvent for silver. In dissolving silver, the standard extra-solution is more and more impoverished in copper, and its permanency is increased until, finally, its solvent power for silver becomes extremely weak.

§ 327. *Solubility of Gold Sulphide.*

Au_2S_3 is readily decomposed by the extra-solution. In the following experiments, 0.500 gm. gold were dissolved in aqua regia,

and precipitated with H_2S. The Au_2S_3 was treated with 1000 c. c. of a standard extra-solution containing 1½ per cent. $CuSO_4+5aq$. After one hour 0.460 gm. gold had gone into solution; after an hour and a half another sample showed nearly all the gold, namely, 0.482 gm., dissolved. In this case, too, copper sulphide is precipitated.

In another series of experiments 1.333 gm. gold were, in each case, converted into Au_2S_3, and treated with extra-solutions of varying concentration, and at different temperatures. The results were not uniform. The solutions acted best at a moderate temperature, and heating them seemed to be of no benefit. Evidently, upon heating the solutions, Au_2S_3 is decomposed to metallic gold, and the latter is less amenable to extra-solution than the sulphide.

§ 328. *Solubility of Silver Arsenate and Antimonate.*

The effect of extra-solution on these silver-salts does not differ from that of ordinary solution. (Russell.)

The Effect of Extra-Solution on Silver-Ores.

We now arrive at the most important part of the experiments—so far as their direct practical value is concerned—the effect of the extra-solution in extracting silver from ores. Regarding the assay-office lixiviation-tests, it will suffice to state, for the present (this subject will be fully treated in a separate chapter), that small quantities of ore, ¼ or ½ A. T., are treated with an excess of ordinary or extra-solution, or with both, and the tailings are assayed. By comparing the value of the ore with the value of the tailings, the percentage of silver extracted is obtained. There are always two lixiviation-tests made, one with ordinary solution only, and one in which extra-solution is used, so that the beneficial effect of the latter becomes apparent.

§ 329. *The Effect of Extra-Solution on Raw Ores.*

Most interesting are the results which Mr. Russell has obtained in lixiviating silver-ores without previous roasting. Their importance can hardly be over-estimated, since thereby an entirely new field is opened to the lixiviation-process. It is most likely that in many cases, where raw amalgamation is practiced with profit,

lixiviation can take its place. There is, however, one class of oxidized ores which may not yield a high percentage of silver if treated by raw lixiviation. If a considerable portion of the silver occurs with lead carbonate, the extra-solution will not extract this part of the silver on account of the insolubility of lead carbonate in hyposulphite solutions.

The ores in the tabulated statement below belong mostly to the so-called " base ores ;" still the percentage of silver extracted is, in many cases, remarkably high.

From this table it appears that, of the sulphuret-ores, those are principally affected by extra-solution which carry the silver in the form of native silver, silver-sulphuret, and the group of antimonial and arsenical sulphurets, like pyrargyrite, stephanite, and polybasite. Being especially familiar with the ores from the Lexington, Manhattan Company's, and Ontario mines, I think a comparison of results will be interesting. The Lexington ore is the basest of the three, and any one seeing this ore in bulk would not suppose that a lixiviation-process could extract 68.5 per cent of its silver without roasting. Much of its silver occurs native, and the principal portion of it is combined with iron pyrites as silver sulphide. Then comes the ore from the Manhattan mines. Native silver here is rare. The principal silver-bearing minerals are ruby-silver, stephanite, and polybasite. Ontario ore shows the lowest result. Native silver occurs, but to much less extent than at the Lexington. About 88 per cent. of its silver is found as fahl-ore, and this mineral seems to be more refractory in contact with the extra-solution.

These tests were made by Mr. Russell in 1883, with ore-samples taken at that time from the above named mines.

The battery-sample from Tombstone is of interest, because it shows that Russell's process will extract from such ores a higher percentage than raw amalgamation. In the annual report—covering the time when these samples were taken—of the Tombstone Mill and Mining Company, the yield of silver by raw amalgamation, is given as 76 per cent. of the assay-value.

Below is another series of raw lixiviation-tests with ores from various mines.

TABLE No. 329. a.

Name of Mine.	Process by which ores are worked at present.	Silver per ton of ore. Ounces.	Silver extracted by Lixiviation with: Ordinary Solution. Per cent.	Silver extracted by Lixiviation with: Extra-solution. Per cent.	Difference. Per cent.	Character of Ore.
Lexington, Montana	Roasting and amalgamation.	54.0	26.0	68.5	42.5	Native silver, silver bearing pyrites of iron, zinc-blende, galena, pyrites of copper.
Manhattan Company, Nevada	Roasting and amalgamation.	164.8	8.3	57.2	48.9	Group of antimonial and arsenical sulphurets of silver, fahl-ore, zinc-blende, galena, pyrites of iron and copper.
Ontario, Utah	Roasting and amalgamation.	90.8	7.0	31.9	24.9	Fahl-ore, native silver, zinc-blende, not much pyrites of iron and copper, galena,
Mount Cory, Nevada	Roasting and lixiviation.	54.0	39.6	63.0	23.4	The same minerals as in Ontario ore, but less base and somewhat decomposed.
		42.5	16.0	40.0	24.0	
Custer, Idaho	Roasting and amalgamation.	30.8	27.8	72.8	45.0	Principally stephanite, pyrites of iron, etc.
Ramshorn, Idaho	Smelting.	89.6	34.4	50.5	16.1	Principally iron carbonate, some lead minerals, pyrites of copper, copper-silver glance.
Jesus Maria, Parral, Mexico	Roasting and amalgamation, or lixiviation.	71.2	16.0	54.5	38.5	Principally pyrites of iron, some galena and zinc-blende, no precious silver min. visible.
Mines near Durango, Mexico	Patio process.	58.0	2.0	39.8	37.8	Completely oxidized, no sulphurets visible.
		14.8	16.9	51.4	34.5	Below water-level the ore contains galena.
		32.8	8.0	65.3	57.3	The oxidized ore, most likely, contains lead sulphate, and carbonate.
		19.2	34.9	70.4	35.5	
Bertrand, Nevada	Roasting and lixiviation.	24.0	8.0	11.8	3.8	Principally lead antimonate.
Tombstone, Arizona	Raw amalg'mat'n. Battery sands.	52.8	70.5	81.1	10.6	Oxidized or free-milling ores, with more or less silver chloride and lead sulphate and carbonate.
	Sands from tailings beds.	9.2	60.9	70.0	9.1	
	Slimes from tailings beds.	17.2	81.4	81.4	none.	
Sierra Grande, New Mexico	Tailings from raw amalgamation, concentrated	67.6	61.1	70.2	9.1	Oxidized or free-milling ores, with silver chloride and lead sulphate and carbonate.
Sombretillo, Sonora, Mexico	?	86.4	75.0	84 0	9.0	Gangue composed of about 30 p. c. quartz and 70 p. c. carbonate of lime and magnesia. Silver, most likely, present as chloride.
Ore from dump of Chrysolite, Leadville, Colorado		8.0	24.0	40.0	16.0	Silver combined with lead sulphate and carb. Also silver chloride.

TABLE No. 329. b.

Name of Mine.	Value of Ore. Ounces p. t.	Silver extracted by Lixiviation with: Extra-solution. Per cent.	Ordinary solution. Per cent.	Difference. Per cent.
Grand Central, Arizona,	43.2	93.7	86.6	7.1
Horn Silver, Utah,	184.0	90.7	81.6	9.1
Price River, Utah,	25.0	87.2	77.6	9.6
Silver Reef, Utah,	45.0	77.8	50.7	27.1
Gray Rock, Montana,	19.0	73.4	47.4	26.0
Pearsall, Colorado,	100.0	68.3	36.6	31.7
Tybo, Nevada,	20.0	58.0	39.0	19.0
Silver Spring, Montana,	62.8	49.0	24.4	24.6

These examples will be sufficient to show the practical importance of the subject.

The ore-samples in the preceding table were all pulverized so as to pass either through a No. 30 or a No. 40 screen. The following figures show the influence of finer crushing.

Sample of Oxidized Ore from the 200-foot Level of the Ontario Mine. Value, 48.8 ounces Silver per ton.

Pulverized so as to pass. No. screen.	Lixiviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.
20	50.0	60.0
30	50.0	65.2
40	52.0	65.6
50	52.5	66.4
60	53.3	67.0
70	54.0	74.6
80	55.0	75.5
Difference between No. 20 and No. 80 screens,	5.0	15.5

Sample of Oxidized Ore from the 400-foot Level of the Ontario Mine. Value, 100.0 ounces Silver per ton.

Pulverized so as to pass. No. screen.	Lixiviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.
40	46.4	72.4
50	46.6	74.6
60	48.0	75.4
70	55.0	78.0
80	55.2	78.4
90	——	79.8
Difference between No. 40 and No. 90 screens,	8.8	7.4

Sample of Ore from the Yedras Mine, Mexico. Value, 33.6 ounces Silver per ton.

Pulverized so as to pass. No. screen.	Lixiviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.
40	45.3	50.6
60	45.3	58.4
90	45.3	64.3
Difference between No. 40 and No. 80 screens,	none	13.7

From these results it appears that fine crushing is with some ores essential to extract a high percentage of the silver. The same is necessary in raw amalgamation with chemicals, where the ore is finely ground in the pan after a more or less coarse crushing in the battery. In both cases the particles of the silver-minerals, on account of their density, are not penetrated by the solution, and the reaction takes place on the surface only. It is quite different with roasted ores. Crushing through a No. 20 or No. 30 screen is, in most cases, sufficient for good roasting, whereby the particles of ore are left in a porous condition. It is well known that the expense and difficulty of crushing increases materially with fineness. The same is the case with lixiviation considered from a mechanical standpoint. Hence, it may be, in such cases, more profitable to crush the ore as coarse as possible, and first roast it before extracting the silver by lixiviation. With well appointed mechanical furnaces, of large capacity, the expense of roasting, so far as labor and fuel are concerned, is slight. The cost of the salt is, in chloridizing-roasting, generally the principal item of expense.

What I have said above about the influence of fine crushing in raw lixiviation does not hold good as a general rule. Mr. Russell found that in the greater number of cases crushing through very fine screens does not improve results materially.

It will be proper to refer here to the fact that lixiviation of raw ores has been practiced before, especially at the Old Telegraph mine, Utah. In the latter case the ore contained silver chloride and lead-minerals. These were concentrated after lixiviation. In attempting to concentrate the ore at once the silver chloride was lost. There is no doubt that the extra-solution would have extracted a much higher percentage of the silver. This system may be used to advantage in many cases where the character of

the ore is such that a large percentage of the silver is lost by direct concentration, and the silver is in a form soluble in extra-solution.

§ 330. *The Effect of Extra-Solution on Ores that have been Oxidized by Roasting.*

In view of the energetic reactions of the extra-solution, I considered it of interest to carry out the idea of lixiviating ores after they had been subjected to an oxidizing-roasting. These experiments were, at first, made on a small scale, that is, by roasting samples in a muffle.

The following table contains the results of Mr. Russell's whole series of experiments. In each case a sample of 500 gm. was roasted.

In reviewing this table it is, in the first place, evident that the ores become less fit for lixiviation the longer they have been roasted. . The two Ontario samples, roasted at different temperatures, show the injurious effect of high heat. If we compare the lixiviation-tests in the above table with those in the previous chapter, made with raw ores and extra-solution, we find that in all cases, excepting the Ontario ore, the roasting proved to be decidedly detrimental. Concerning the loss of silver, experienced in muffle-roasting, Mr. Russell's figures are entirely in accordance with Plattner's observations on this subject. The loss of silver, other conditions being equal, is a function of temperature, time, and character of the ore. How very marked the influence of time is, these experiments plainly indicate. Hence, the instantaneous roasting, as it is done in a Stetefeldt furnace, must reduce that influence to a minimum.

The curious fact, that in oxidizing-roasting of silver ores, the percentage of silver extracted by hyposulphite solutions decreases the more the roasting is prolonged, led to the question: What will be the result if the roasting is done in the shortest possible time, namely, in a Stetefeldt furnace? This experiment was carried out by Mr. Russell at the Ontario Mill. After cleaning out the chloridized ore from the hoppers of the shaft and flue, the feeding of the salt was suspended for two hours. Samples of raw ore, taken every half hour from the feeder, showed the pulp to contain

TABLE No. 330. a.

Ore from	Temperature.	Time. hours.	Per cent. of silver extracted with: Ordinary solution.	Per cent. of silver extracted with: Extra-solution.	Per cent. of silver lost in roasting.
Ontario Mine.	Cherry-red.	½	31.1	55.8	2.2
		1	28.6	40.0	8.8
		1½	19.3	28.9	10.4
		2	17.0	26.9	11.0
		2½	17.3	23.0	13.4
		3	14.1	11.0	17.6
Ontario Mine.	Dark-red.	½	44.2	64.5	0.7
		1	35.1	44.5	7.0
		1½	31.0	34.8	7.1
		2	24.6	26.4	8.7
		2½	23.8	26.8	8.3
		3	37.4(?)	32.3	8.3
Ramshorn Mine.	Dark-red.	½	13.4	17.4	2.3
		1	7.7	11.6	5.0
		1½	7.0	7.6	5.3
		2	6.6	7.7	5.4
		2½	3.9	5.5	5.8
		3	2.9	2.9	5.8
Manhattan Mine.	Dark-red.	½	10.0	22.2	4.8
		1	5.0	18.1	6.5
		1½	8.0	13.0	12.4
		2	2.6	....	13.0
		2½	1.0	2.0	14.0
		3	....	....	14.8
Lexington Mine.	Dark-red.	½	17.6	32.5	3.2
		1	6.2	12.0	3.7
		1½	6.6	6.6	4.0
		2	1.2	4.7	4.3
		2½	2.0	5.5	4.6
		3	....	....	4.8
Bertrand Mine.	Dark-red.	½	1.0	6.0	From 4.3 to 5.7 p. ct.
		1	....	3.7	
		1½	....	....	
		2	1.3	2.0	
		2½	2.4	1.7	
		3	1.6'	1.3	
Custer Mine.	Dark-red.	½	27.0	31.5	From 1 to 2.1 p. ct.
		1	22.5	29.8	
		1½	26.0	30.0	
		2	21.0	29.8	
		2½	29.0	28.4	
		3	19.0	25.0	

only a fraction of 1 per cent. in salt, the latter being derived from pulp lodged in the troughs of the conveyor-screws. The roasting was conducted at a very high temperature, and with an abundant supply of air for oxidation. Samples of roasted ore were taken from the top of the pile in the shaft and flue, and lixiviated.

Sample from.	Silver extracted with: Ordinary solution. Per cent.	Extra-solution. Per cent.
Shaft	12.8	55.9
Shaft	19.0	58.6
Shaft	28.8	55.6
Flue	14.4	45.5
Flue	39.9	67.3
Flue	25.0	37.0

At the expiration of two hours, the ore was discharged from the shaft and flue, allowed to remain in a pile on the cooling-floor for twelve hours, and carefully sampled. Below are the results of lixiviation.

Silver extracted by lixiviation with:	Ore from: Shaft. Per cent.	Flue. Per cent.
Water	17.5	4.9
Water and H_2SO_4	16.3	5.2
Ordinary solution	86.0	62.4
Extra-solution	89.7	69.2

These results are so interesting and important, and so contrary to accepted metallurgical principles, that it was considered expedient to repeat the experiments.

The following five experiments were made with Ontario ore which carried an unusually large percentage of zinc-blende. As in the previous case, a fraction of 1 per cent. of salt (from three-tenths to six-tenths) remained in the ore. If we call the amount of fire generally used in chloridizing-roasting at the Ontario Mill "normal," this condition was only maintained in experiment No. 4. In all other cases extra fire was used, especially in Nos. 1, 2, and 3. The final results, that is, after the ore has remained on the cooling-floor, indicate that this extra fire is beneficial. In experiment No. 3 the admission of air was materially reduced, and the temperature was highest. The initial result in this case is the lowest of all, but the increase in per cent. of silver extracted, after the ore has remained on the cooling-floor, is by far the highest, and the final result the best. When the ore in experiment No. 4 was

discharged, it was not red-hot. Although the initial result is much better than in No. 3, the gain by remaining on the cooling-floor is less than one-half of No. 3.

TABLE OF EXPERIMENTS. No. 330. b.

Samples taken from inside of shaft and flue.	Experiment.				
	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.
Value of raw ore, oz. per ton	74.8	83.2	82.4	102.8	79.5
Value of roasted ore, shaft, oz.	66.	68.	62.	92.	72.
Value of roasted ore, flue, oz.	94.	84.	84.	108.	91.
Per cent. of salts soluble in water { shaft	6.5	3.0	3.5	3.2	4.2
Per cent. of salts soluble in water { flue	2.0	1.5	1.5	5.0	2.1
Per cent. of salts soluble in sodium hyposulphite { shaft	14.0	7.5	9.5	10.7	9.3
Per cent. of salts soluble in sodium hyposulphite { flue	9.0	5.0	6.5	7.0	6.0
Per cent. of silver extracted by leaching with water { shaft	5.2	5.3	5.8	4.8	5.6
Per cent. of silver extracted by leaching with water { flue	12.0	----	----	4.0	----
Per cent. of silver extracted by ordinary solution { shaft	30.9	20.9	31.5	23.0	27.5
Per cent. of silver extracted by ordinary solution { flue	29.0	26.9	15.0	25.2	18.8
Per cent. of silver extracted by extra-solution { shaft	66.5	60.6	50.0	59.6	66.6
Per cent. of silver extracted by extra-solution { flue	61.6	70.9	45.5	66.3	67.8

Samples taken after the ore has remained 16 hrs. on cooling-floor.	Composition of charge, about, parts.				
	3 shaft, 2 flue.	3 shaft, 2 flue.	Shaft only.	4 shaft, 1 flue.	Shaft only.
Per ct. of salts soluble in water	8.5	5.0	6.7	5.5	5.5
Per ct. of salts sol. in sod. hypos.	13.2	9.5	12.7	11.0	10.0
Per ct. silver extracted by water	2.0	----	12.6	6.8	10.0
Per cent. of silver, ordinary sol.	74.2	64.8	83.4	35.3	73.7
Per cent. of silver, extra-solution	84.0	80.0	85.8	77.6	81.9
Increase of per cent. in silver extracted by extra-solution as compared with the samples taken from inside of shaft and flue, considering the composition of charge.	19.5	15.3	35.8	16.7	15.2
Per cent. of silver extracted by lixiviating 2 tons.	85.0	72.1	85.7	74.4	80.6

The first question which presents itself in reviewing this interesting table is: In what condition does the silver exist in the roasted ore? The silver soluble in water is, undoubtedly present as silver sulphate. From the fact that a large percentage of silver is extracted by ordinary solution, especially considering the final results, it follows that the principal part of the silver must be in combinations equally soluble in ordinary and extra-solution. As such, we have found the arsenates and antimonates of silver. Nearly all the silver in Ontario ore occurs as fahl-ore, and from it these combinations are derived. Some silver, it is fair to presume,

is present in metallic form, and this is easily soluble in extra-solution. Silver sulphide as such does not occur in the ore, and supposing it to be present, is so easily oxidized that it can hardly exist after roasting.

The solubility-tests of roasted ore in water, show a marked difference between shaft and flue. In chloridizing-roasting, in the Stetefeldt furnace, the formation of sulphates takes place principally in the flue. The salt converts the sulphates of copper and zinc completely into chlorides in the shaft, but not in the flue. The larger percentage of sulphates formed in oxidizing-roasting in the shaft of the Stetefeldt furnace, must be due to the greater coarseness of the ore which is here deposited. It will be seen from the difference in value of the roasted ore in the shaft and flue, that the latter contains much more silver-bearing mineral, and that in consequence more sulphates might be expected in this portion of the ore. After the roasted ore has remained red-hot for a considerable time on the cooling-floor, the quantity of salts soluble in water and sodium hyposulphite has materially increased. The sulphates of copper, zinc, and manganese react most energetically on the silver, increasing the quantity of silver sulphate, antimonate and arsenate, thus converting it into combinations soluble in hyposulphite solutions.

From the facts recorded about the influence of time in muffle-roasting, it appears that a momentary roasting is essential to the success of the process.

In other words, the longer the ore has been exposed to the influence of external heat in the process of oxidation, the less soluble become the silver-compounds in hyposulphite solutions. It is doubtful whether instantaneous roasting produces silver-combinations differing from those formed in reverberatory furnace roasting, and the difference in solubility caused thereby; or is the result in reverberatory furnace roasting due merely to the long exposure of the ore to the flame? As to the latter cause, analogous phenomena are not wanting in chemistry. I only need allude to the peculiarity of some metallic oxides becoming less soluble in acids after they have been exposed to a high temperature for a long time.

I hope these experiments will be repeated with other ores, and in other mills. An obscure critic has said, that on account of the

small percentage of salt left in the ore (from $\frac{3}{10}$ to $\frac{6}{10}$ per cent.), this was not oxidizing-roasting pure and simple; that the salt left in the ore was responsible for the results. With Ontario ore, requiring not less than 15 per cent. salt for a high chlorination of the silver, this argument falls to the ground. On the contrary, I am inclined to believe that this small amount of salt had a damaging effect, if any. By roasting Ontario ore with 2 per cent. salt, only 44.5 per cent. of its silver could be extracted by lixiviation. The silver sulphate in the roasted ore shows that the chlorine in the salt never reached the silver, but combined, for a moment, with base metals, and then escaped from the furnace as hydrochloric acid before the ore dropped to the bottom of the shaft.

I can not end this discussion without pointing out the practical importance of these experiments. If it is possible to extract silver by lixiviation without using salt in roasting, we not only save the expense of the salt, but also the cost of drying and crushing it, and of the plant needed for these purposes. For localities where salt is very costly, and the ores are of low grade, a decrease in the yield of silver, of several per cent., would still leave a balance in favor of oxidizing-roasting.

The Effect of Extra-Solution on Ores that have been subjected to a Chloridizing-Roasting.

We have to make here a distinction between roasted ores free from caustic lime, and those containing it in a perceptible degree. The former are first lixiviated with ordinary solution to extract, principally, the silver chloride, and then the extra-solution is applied to dissolve silver-compounds not soluble in ordinary solution. If roasted ores containing caustic lime are treated in the same way, the extra-solution fails to work properly; but if it is applied at once, the cuprous hyposulphite neutralizes and counteracts the damaging effect of the caustic lime, and leaves the silver in a soluble form. An exception to this rule will be noted later.

§ 331. *A. Roasted Ores free from Caustic Lime.*

Lixiviation-Tests at the Ontario Mill.—Mr. Russell made daily, lixiviation-tests of the Ontario mill-samples both with ordinary and extra-solution, and compared the results with those obtained by amalgamation.

The Ontario ore is roasted with salt, in two Stetefeldt furnaces. After discharging the ore from the furnaces, it is piled up on the cooling-floor, and left for 12 hours undisturbed. It is then cooled with a spray of water, and loaded into cars for charging the pans. From these cars the samples were taken to which reference is made below. The percentage of silver extracted from the roasted ore by amalgamation, is calculated from the silver remaining in the tailings, by comparing their value with the value of the roasted ore after leaching out its soluble salts with water.

TABLE No. 331. a.

Monthly averages of lixiviation-tests, and percentage of silver extracted by amalgamation, at the Ontario Mill.

Samples taken:	Lixiviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.	Silver extracted by amalgamation. Per cent.
March, 1882	84.6	92.0	89.6
April, "	87.6	91.3	89.9
May, "	88.5	92.9	92.1
June, "	87.5	93.1	91.5
July, "	91.2	93.0	91.8
August, "	89.8	92.5	90.6
October "	87.3	92.7	89.7
November, 1882	89.2	93.1	90.4
December, "	88.5	92.9	89.2
March, April, 1883	87.3	92.7	89.7
Averages	88.1	92.6	90.4
Differences	4.5	2.2	

The effect of extra-solution on Ontario ore, if not well chloridized.—Much more pronounced becomes the difference between the dissolving energy of the two solutions if ore is lixiviated which does not show a high chlorination by the ordinary method. Experience has established the fact that the Ontario ore, and all ores with a large percentage of sulphurets, especially zinc-blende, considerable time is required to complete the chlorination of the silver. This is effected by leaving the ore in a red-hot state for several hours. There are ores which do not necessarily require this treatment, but in all cases the effect is more or less beneficial. If samples of Ontario ore are taken immediately after a charge has been drawn from the furnace, the percentage of the silver chloridized is very low, especially if the ore has been roasted at a

moderate temperature. The dust seems to make an exception to this rule, but, in fact, it does not, if we consider that it remains in the dust-chambers for a long time before it is discharged.

TABLE No. 331. b.

Samples of Ontario ore lixiviated immediately after discharging from the Stetefeldt Furnace.

From where discharged:	Liviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.	Differences: Per cent.
Shaft	60.7	86.1	25.4
Return-flue	61.8	84.2	22.4
Dust-chambers	90.7	93.3	2.6
Shaft	65.2	93.2	28.0
Return-flue	60.7	91.8	31.1
Dust-chambers	80.0	91.4	11.4
Shaft	60.5	90.0	29.5
Shaft	65.4	93.9	28.5

These results are very important—showing how completely the extra-solution corrects imperfection in chloridizing-roasting.

TABLE No. 331. c.

Lixiviation-tests made with Roasted Ores from Various Mills.

Furnace Used.	Name of Mill.	Ordinary Solution. Per cent.	Extra Solution. Per cent.	Difference. Per cent.
Howell	Alice, Montana	90.0	93.7	3.7
"	" "	73.7	89.2	15.5
"	" "	86.0	92.0	6.0
Bruckner	Custer, Idaho	83.3	91.5	3.2
Stetefeldt	Lexington, Montana,	92.4	93.6	1.2
Stetefeldt	Manhattan, Austin	94.1	94.6	0.5
Howell	Black Warrior, Arizona	94.0	95.2	1.2
Bruckner	Bertrand, Nevada, Mt. Cory Ore	89.5	95.1	5.6

Whenever the chlorination of the silver has been carried to a very high percentage, it cannot be expected that the results of lixiviation with the two solutions should differ very much. Hence, it will be necessary to investigate in every special case how much profit can be derived from the use of the extra-solution.

§ 332. *B. Roasted Ores containing Caustic Lime.*

In lixiviating roasted ores containing caustic lime, the difference in extraction of silver between ordinary and extra-solution is much

more pronounced. As a prominent example, I cite some results from the Yedras Mill, Mexico. The ore contains from 50 to 60 ounces silver per ton.

TABLE No. 332.

Lixiviation-Tests made with Roasted Yedras Ore.

Furnace Used.	Per cent. Silver extracted.		Difference. Per cent.
	Ordinary Solution.	Extra-Solution.	
Reverberatory	77.9	90.0	12.1
"	66.8	91.0	24.2
"	48.6	78.5	29.9
Brueckner	38.3	78.3	40.0

Further information on this subject will be found in Chapter XVIII.

§ 333. *Effect of Extra-Solution on Tailings.*

It is generally not known that tailings from ores that have been amalgamated, either raw or after roasting, contain a considerable portion of their silver in combinations soluble in hyposulphite solutions. Hence, it may be profitable to lixiviate such tailings, unless they are in the form of slimes, and offer insurmountable difficulties to filtration. The same applies to tailings from the ordinary lixiviation-process from which the extra-solution will extract more silver. Mr. Russell tested a great number of tailings-samples from the Ontario Mill, finding that from 20 to 50 per cent. of the silver could be lixiviated. I obtained similar results with tailings from the Lexington Mill. Below is more evidence of the same character.

TABLE No. 333.

Lixiviation-tests made with Tailings from various Mills.

Name of Mill.	Process.	Silver extracted by: Ordinary Solution. Per cent.	Extra-Solution. Per cent.	Difference. Per cent.
Sierra Grande, New Mex.	Raw amalgamation	61.1	70.2	9.1
Black Warrior, Arizona	Roasting and amalgamation	57.5	62.5	5.0
Bertrand, Nevada	Roasting and ord. lixiviation	7.2	59.8	52.6
Raymond and Ely, Nev.	Raw amalgamation	30.4	42.4	12.0
Custer, Nevada	Roasting and amalgamation	17.0	31.0	14.0
Belmont, Nevada	" " "	22.8	28.0	5.2
Manhattan, Nevada	" " "	23.6	25.3	1.7

CHAPTER IV.

The Extraction of Gold.

The problem of extracting from gold-bearing silver-ores a high percentage of both precious metals by the application of one humid process, has not yet been solved. If such ores are suitable to raw amalgamation in pans, the yield in gold is often quite satisfactory, while that in silver remains low. Chloridizing-roasting improves results so far as silver is concerned, but diminishes the yield in gold. Although the lixiviation-process with hyposulphite solution is not adapted to working gold-ores proper, and no one would think of using it for that purpose, it enters into successful competition with amalgamation in beneficiating gold-bearing silver-ores, after they have been roasted with salt. Mr. Russell made a number of tests with ore from the Lexington Mill. The ore carried from 42 to 50 ounces silver and from 0.5 to 0.9 ounces gold, per ton, and had been chloridized in a Stetefeldt furnace. Lixiviation extracted from 60 to 70 per cent. of the gold. The statistics of the Lexington Mill, at that time, recorded a yield in gold of 50 to 53 per cent. by amalgamation. Mr. R. D. Clark worked roasted ore from the Alice Mine, at the Bértrand Mill, by lixiviation, and also reported a better extraction of the gold than that obtained at the Alice Mill by amalgamation. I found that the percentage of gold extracted by amalgamation decreases with the increase of temperature at which the roasting is conducted, and with the time consumed in roasting, or the time during which the charge is kept red-hot, after being withdrawn from the furnace. Unfortunately in most cases, amalgamation requires for a high extraction of the silver, roasting at a highly elevated temperature, and keeping the charge red-hot on the cooling-floor for several hours.

Such conditions are not needed if the ore is lixiviated by the extra-solution of the Russell process. Hence, a quick roasting in a Stetefeldt furnace may leave the ore in a condition most favorable for a high extraction of both silver and gold.

This subject requires further investigation. Reliable modern statistics are lacking.

CHAPTER V.

Chemistry of the Wash-Water.

§ 501. *The Composition of Roasted Ores.*

Ores that have been subjected to a chloridizing-roasting contain a great number of salts soluble either in water or in brine. The salts soluble in water are, principally, sodium sulphate and chloride, a large portion of the latter remaining undecomposed in roasting. Besides, there may be the sulphates of manganese, zinc, copper, iron, aluminium, and magnesium; the chlorides of the same metals, and of calcium, barium, and strontium, the latter being of rare occurrence. The chlorides of barium and strontium, however, will be immediately decomposed on solution, and insoluble sulphates of these metals will be precipitated. The same reaction would take place in the presence of the easily soluble barium hydrate. Finally, there may be sodium arsenate. Salts not so easily soluble in water, as those mentioned above, are—cuprous chloride, lead chloride, calcium sulphate, sodium antimonate, and caustic lime. Lead chloride, if dissolved, can not remain as such, but will be precipitated as lead sulphate. Silver chloride, lead sulphate and antimonate, are almost insoluble in water, but more or less soluble in concentrated leaching-brine. In the same way the solubility of cuprous chloride, calcium sulphate, and caustic lime is increased by the presence of sodium chloride, and other salts, in water. If caustic lime exists in the ore in sufficient quantity, the soluble salts of the base metals are decomposed with precipitation of hydroxides.

Since all roasted ores have to be leached with water, prior to the application of the hyposulphite solutions, a considerable percentage of the silver in the ore may be taken up by the so-called first wash-water.

§ 502. *Necessity of Leaching.*

Leaching with water is necessary:

1st. So as not to overload the lixiviation-solution with foreign salts.

2nd. Because the sulphides, precipitated from the lixiviation-solution, would become very base if the salts of the base metals were not removed, as much as possible, prior to the extraction of the silver.

It is of importance to know the solubility of the principal salts which come into play in base-metal leaching; hence, this subject will be considered in detail.

A. Salts Easily Soluble in Water.

§ 503. *Solubility of Na_2SO_4.*

1000 c. c. water dissolve:

At 10° C., or 50° F., 230.4 gm. $Na_2SO_4 + 10aq.$
" 20° C., or 68° F., 527.6 " Na_2SO_4.
" 50° C., or 122° F., 468.2 " "
" 100° C., or 212° F., 426.5 " "

§ 504. *Solubility of NaCl.*

1000 c. c. water dissolve:

At 0° C., or 32° F., 355.2 gm. NaCl.
" 14° C., or 57.2° F., 358.7 "
" 25° C., or 77° F., 361.3 "
" 50° C., or 122° F., 369.8 "
" 100° C., or 212° F., 396.1 "

§ 505. *Solubility of $CaCl_2$.*

1000 c. c. water dissolve:

At 10° C., or 50° F., 633.5 gm. $CaCl_2$.
" 40° C., or 104° F., 1204.8 "
" 100° C., or 212° F., 1550.0 "

§ 506. *The Sulphates and Chlorides of Manganese, Zinc, Copper ($CuCl_2$), Iron, Aluminium and Magnesium.*

These are all easily soluble in cold and in hot water. Sodium arsenate is also soluble. The above named salts exist in roasted ores in comparatively small quantities so that a detailed account of their solubilities is not of much importance.

B. Salts not freely soluble in Water.

§ 507. *Solubility of Cu_2Cl_2.*

1000 c. c. water dissolve:

At 14° C., or 57.2° F., 0.9 gm. Cu_2Cl_2.
" 100° C., or 212° F., 1.35 "

§ 508. *Solubility of $PbCl_2$.*

1000 c. c. water dissolve:

At 12.5° C., or 54.5° F., 7.4 gm. $PbCl_2$.
" 100° C., or 212° F., 50.0 "

§ 509. *Solubility of $CaSO_4 + 2aq$.*

1000 c. c. water dissolve:

At 0° C., or 32° F., 1.90 gm. $CaSO_4 + 2aq$.
" 18° C., or 64.4° F., 2.05 "
" 36° C., or 96.8° F., 2.14 "
" 53° C., or 127.4° F., 2.11 "
" 99° C., or 210.2° F., 1.75 "

§ 510. *Solubility of CaO.*

The solubility of CaO in water decreases with increase in temperature. Solubility-coefficients, given by different authorities, do not agree very closely. According to A. Lamy's investigations, the solubility is influenced by the method of preparing CaO. For CaO obtained by calcining pure marble, Lamy gives the following figures:

1000 c. c. of a saturated solution contain:

At 0° C., or 32° F., 1.381 gm. CaO.
" 15° C., or 50° F., 1 299 "
" 30° C., or 86° F., 1.162 "
" 60° C., or 140° F., 0.868 "
" 100° C., or 212° F., 0.576 "

According to Dalton:

1 gm. CaO, at 15° C., requires 778 c. c. water, and 1 gm. CaO, at 100° C., requires 1270 c. c. water for solution.

Bineau gives the following solubilities:

1 gm. CaO, at 18° C., or 64.4° F. requires 780 c. c.
" " 100° C., or 212° F. " 1500 c. c.

water for solution.

§ 511. *Solubility of Sodium Antimonate.*

1000 c. c. of boiling water dissolve, 2.86 gm. $2NaSbO_3+7aq.$, or 1.73 gm. $NaSbO_3$. The solubility in cold water is much less.

C. Salts not easily soluble in Water, but more soluble in Brine.

§ 512. *Solubility of Cu_2Cl_2.*

1000 c. c. concentrated brine dissolve:

At 40° C., or 104° F., 80.0 gm. Cu_2Cl_2.
" 90° C., or 194° F., 160.0 "

1000 c. c. brine, with 15 per cent. NaCl, dissolve:

At 14° C., or 57.2° F., 35.0 gm. Cu_2Cl_2.
" 84° C., or 183.2° F., 60.0 "
" 90° C., or 194° F., 100.0 "

1000 c. c. brine, with 5 per cent. NaCl, dissolve:

At 40° C., or 104° F., 14.0 gm. Cu_2Cl_2.
" 90° C., or 194° F., 26.0 "

These figures are, according to T. Sterry Hunt, approximate only. Both on cooling and diluting these solutions with cold water, Cu_2Cl_2 is precipitated.

§ 513. *Solubility of $PbCl_2$.*

1000 c. c. concentrated brine dissolve:

At about 15° C., or 50° F., 10.8 gm. $PbCl_2$.
At boiling point, 53.7 "

§ 514. *Solubility of $CaSO_4 + 2aq.$*

According to Anthon, 1000 c. c. concentrated brine dissolve, at ordinary temperature, 8.2 gm. $CaSO_4+2aq.$; hence, for 1000 gm. NaCl present in the solution, 25 gm. $CaSO_4+2aq.$ are dissolved.

The analysis of a brine, saturated with gypsum, from Koesen, Germany, shows for 8.52 per cent. NaCl, 0.63 per cent. $CaSO_4$+2aq. in solution, besides fractions of one per cent. of the sulphates of sodium, potassium, and magnesium; 1000 c. c. of this brine contain, at ordinary temperature, 6.3 gm. $CaSO_4$+2aq., or per 1000 gm. NaCl present, 74 gm. $CaSO_4$+2aq. are in solution. This would indicate that diluted brine is comparatively a better solvent for gypsum than concentrated brine even if we deduct the quantity of gypsum dissolved by water alone. I have found no data as to influence of temperature, but judging by analogy from the solubility of gypsum in water, and in sodium hyposulphite solutions, I assume that its solubility in brine is diminished by increase of temperature.

D. Salts practically insoluble in Water, but soluble in Brine, and in solutions of other Chlorides.

In this respect I have found data for silver chloride only.

§ 515. *Solubility of AgCl in Brine.*

According to Frémy and Pelouze, the quantity of AgCl dissolved in brine increases in proportion to the units of NaCl present in the solution, and, for solutions of equal concentration, rises with elevation of temperature as follows:

At 0° C., or 32° F., only traces of AgCl are dissolved.

At	10° C., or	50° F.,	1.7 gm. AgCl	per 1000 gm.	NaCl.
"	18° C., or	64.4° F.,	2.4 "	"	"
"	100° C., or	212° F.,	6.8 "	"	"

It is not stated whether these chemists employed saturated solutions or not. Vogel found that 1000 c. c. concentrated brine dissolved, at ordinary temperature, 0.95 gm. AgCl. If Vogel operated at a temperature of about 18° C., his result would give per 1000 gm. NaCl in solution, 3.0 gm. AgCl dissolved, a value greater than that quoted by Frémy and Pelouze. This difference may be accounted for if Vogel's solubility-coefficient refers to a higher temperature than 18° C.

Greatly at variance with the above figures are the observations of H. Hahn, who found that a solution of sodium chloride, con-

taining 25.96 per cent. NaCl, and of 1.2053 specific gravity, dissolved, at 19.6° C., per 1000 c. c., 1.269 gm. AgCl, or 4.05 gm. AgCl per 1000 gm. NaCl. The difference between Hahn and Vogel may be accounted for by difference in temperature; that between Hahn, and Frémy and Pelouze, by the latter not having used concentrated solutions. The solubility of AgCl in brine is evidently a complex function. It increases for solutions of equal concentration with a rise in temperature, and also increases at equal temperatures for solutions holding equal amounts of NaCl, with the concentration of the solution. The latter view is supported by observations of Fresenius.

§ 516. *Solubility of AgCl in solutions of various Chlorides.*

In this respect, the observations of different chemists show, apparently, still greater discrepancies than those noted above.

Concentrated solutions of $CaCl_2$, $MgCl_2$, and $BaCl_2$, prepared by Vogel at ordinary temperature, and by Hahn at 24.5° C., or 76.1° F., but used at 30.6° C., or 87° F., dissolved per 1000 c. c.:

Solution.	Vogel.	Hahn.
$CaCl_2$	0.930 gm.	8.34 gm. AgCl.
$MgCl_2$	1.710 "	7.09 "
$BaCl_2$	0.143 "	0.74 "

This would give, 13.8 gm. AgCl, dissolved per 1000 gm. $CaCl_2$; 14.6 gm. AgCl, dissolved per 1000 gm. $MgCl_2$, according to details stated by Hahn.

To unravel the mystery as to $CaCl_2$, Mr. E. Jackson, according to Percy, prepared a concentrated solution of $CaCl_2$ at ordinary temperature, added a surplus of AgCl, and found per 1000 c. c. of the solution dissolved:

After 2 days	0.714 gm. AgCl.
" 5 "	0.765 "
" 10 "	1.000 "
" 21 "	1.506 "
" 60 "	1.894 "

These results agree much better with those of Vogel than with Hahn's. Knowing Mr. H. Hahn to be an able and painstaking chemist, I consider his results perfectly reliable, and believe that the apparent discrepancy between his results and those of Vogel

and Jackson, can easily be explained. Neither Vogel nor Jackson state at what temperature their solutions were saturated with $CaCl_2$, or used for dissolving AgCl. Now, the solubility of $CaCl_2$, $BaCl_2$, and $MgCl_2$, increases rapidly with a rise in temperature, and the solvent power of these solutions for AgCl also increases both with their concentration and elevation of temperature. Hence, without knowing the temperature at which Vogel and Jackson operated, their results are not comparable with those of Hahn, whose higher solubility-coefficients must be due both to a higher temperature and concentration of the solutions. Chemists would do well to abolish that indefinite expression "ordinary temperature."

According to Hahn, the chlorides of manganese, iron, copper, and zinc are also good solvents for AgCl. The solutions were saturated at 24.5° C., or 76.1° F.

1000 c. c.	$MnCl_2$	solution dissolved	2.956 gm.	AgCl.
"	$FeCl_2$	"	2.393	"
"	$CuCl_2$	"	0.833	"
"	$ZnCl_2$	"	0.215	"

Fresenius, speaking of the solubility of silver chloride in hot, concentrated solutions of the chlorides of sodium, potassium, and calcium, says: "On sufficient dilution with cold water, the dissolved silver chloride separates so completely, that the filtrate is not colored by sulphuretted hydrogen."

The following solubility-tests were made by the author with leaching-brine as it may be obtained in applying the first wash-water to roasted ores.

It is assumed, in one case, that, in roasting a base ore, free from lime, $\frac{1}{8}$ of the salt remains undecomposed, while $\frac{7}{8}$ are converted into sodium sulphate; and that the first concentrated portion of the wash-water holds per 1000 c. c., 80 gm. NaCl, and 195 gm. Na_2SO_4. In another case, a solution of the same character was saturated with caustic lime and gypsum, or with gypsum alone, to represent a leaching-brine resulting from a roasted base ore containing lime.

E. Salts insoluble in Water, but soluble in Leaching-Brine.

§ 517. *Solubility of Silver Chloride.*

1000 c. c. leaching-brine dissolve:

At 20° C., or 68° F.	0.036 gm. AgCl.
" 75° C., or 167° F.	0.172 "

1000 c. c. leaching-brine, saturated with gypsum, dissolve:

At 20° C., or 68° F.	0.039 gm. AgCl.
" 75° C., or 167° F.	0.166 "

1000 c. c. leaching-brine, saturated with gypsum and caustic lime, dissolve:

At 20° C., or 68° F.	0.036 gm. AgCl.
" 75° C., or 167° F.	0.166 "

§ 518. *Solubility of Silver Arsenate and Antimonate.*

The solubility of silver arsenate and antimonate in brine is the same as that of silver chloride, so far as the amount of silver dissolved is concerned. If precipitates of the former salts are added to brine, they are decomposed with formation of silver chloride. In a solution of silver chloride in brine, potassium arsenate does not produce a precipitate; and upon diluting such a solution with cold water, silver chloride separates, and not silver arsenate.

In treating roasted ores containing silver arsenate and antimonate, the conversion of these salts into silver chloride in contact with brine is evidently very incomplete in consequence of their fused condition. I draw this conclusion from the fact that in pan-amalgamation silver arsenate and antimonate remain in the tailings.

§ 519. *Solubility of Lead Sulphate.*

1000 c.c. leaching-brine dissolve:

At 20° C., or 68° F.	0.092 gm. $PbSO_4$.
" 75° C., or 167° F.	0.192 " "

1000 c.c. leaching-brine, saturated with gypsum, dissolve:

At 20° C., or 68° F.	0.073 gm. $PbSO_4$.
" 75° C., or 167° F.	0.440 " "

1000 c.c. leaching-brine, saturated with gypsum and caustic lime, dissolve:

At 20° C., or 68° F. 0.393 gm. $PbSO_4$.
" 75° C., or 167° F. 0.476 " "

§ 520. *Solubility of Lead Antimonate.*

The solubility of lead antimonate in brine, so far as the quantity of lead dissolved is concerned, is the same as that of lead sulphate, the latter salt going into solution after decomposition of the lead antimonate.

The results of solubilities in the last two paragraphs should not be considered as a correct measure of the quantities of silver and lead that may be taken up by the wash-water. Their importance lies principally in this, that they show a great difference in the effect of leaching with cold and with hot water. In operating on a large scale, the conditions are entirely different, and the character of the solutions is much more complex on account of the presence of many other salts, besides sodium chloride and sulphate.

§ 521. *Conclusions.*

With a knowledge of these facts, the question whether it is more advantageous to use cold or hot water in base-metal leaching, is easily decided. The salts of group A.: Na_2SO_4, NaCl, $CaCl_2$, and other sulphates and chlorides, are so easily soluble, both in cold and hot water, that not much benefit is derived by the use of the latter. In group B., $CaSO_4+2aq.$ and CaO are more soluble in cold than in hot water; hence, preference should be given to the former. Cu_2Cl_2 is less soluble in cold than in hot water; but leaving Cu_2Cl_2 in the charge is beneficial, because it forms extra-solution in subsequent lixiviation with hyposulphite solutions. The same argument holds good for Cu_2Cl_2 as considered in group C. Finally, AgCl being very much less soluble in cold than in hot brine, cold water is best used on that account also. With exception of lead sulphate, the evidence is altogether in favor of cold water; but even a hot leaching-brine does not dissolve much lead sulphate. If roasted ore contains an excess of caustic lime, the caustic leaching-brine cannot hold in solution copper, iron, and manganese;

but it may contain zinc, lead, and alumina, besides antimony and arsenic. Caustic lime does not prevent or materially influence the solution of silver chloride in brine. If the leaching-brine contains calcium chloride, its solvent power for silver chloride will be materially increased. Calcium chloride, however, can only be present in the solution if it exists in excess of sodium sulphate, since, by mixing the two salts, gypsum is formed, which is partly dissolved by brine. A solution of gypsum in brine is, as we have seen, not a better solvent for silver chloride than brine itself.

The first, concentrated brine, which leaves the lixiviation-vats in base-metal leaching, if diluted with sufficient cold water, becomes turbid, and AgCl, Cu_2Cl_2, and $PbSO_4$, besides basic salts of the base metals, are precipitated.

How silver and copper can be recovered from the wash-water, will be shown in § 1404.

CHAPTER VI.

The Chemistry of Sodium and Calcium Sulphide.

The only reagents used for precipitation of silver from hyposulphite solutions, are the sulphides of sodium and calcium. Since the introduction of the latter by Kiss, the majority of metallurgists have tenaciously adhered to this practice, evidently, without examining the subject critically. The necessity of using sodium sulphide exclusively in the Russell process, whenever lead is to be precipitated separately, makes it important to examine into the merits of the two reagents. In fact, the issue is a double one; 1st. Is calcium sulphide preferable as a precipitant for silver? 2d. Is a calcium hyposulphite lixiviation-solution superior to one of the sodium salt?

In considering the first question the following points are involved:

A. The preparation of the sulphide solutions, and their composition.

B. Their action and value as precipitants.

A. Preparation of Sodium and Calcium Sulphide Solutions, and their Composition.

§ 601. *Mode of Preparation.*

Both reagents are prepared by boiling either caustic soda or caustic lime with sulphur. To obtain calcium sulphide, sufficient sulphur must be added to the caustic lime so as to form CaS_4, because the lower calcium sulphides are not easily soluble in water. In fact, a tendency prevails to form CaS_4, even if an excess of lime is present, while an excess of sulphur produces CaS_5. From the slight solubility of calcium hydrate in water, it follows that this process must require considerable time, and that, at the end of the operation, a solution of only moderate concentration must result. Sodium hydrate, on the contrary, is very easily soluble in water, and the sodium monosulphide and all of the polysulphides being equally so, the process must be completed very rapidly.

Hence, not only solutions of any desired concentration may be obtained, but also the lower sulphides of sodium, whereby the consumption of sulphur is reduced to a minimum.

§ 602. *Formation of Hyposulphite Salts.*

Another question, however, is of equal if not of greater importance. The reactions taking place in boiling a caustic alkali with sulphur are expressed by the formula:

$$3RH_2O_2 + (2+2x)S = 2RS_x + RS_2O_3 + 3H_2O.$$

That is to say: For two equivalents of the alkaline sulphide, one equivalent of a hyposulphite salt is formed. Now, let us consider the peculiarities of solutions at boiling-point, containing calcium or sodium hyposulphite. A solution of sodium hyposulphite can be heated to boiling-point without much injury—it decomposes very slowly. A solution of calcium hyposulphite, on the contrary, decomposes rapidly into gypsum and sulphur if heated above 60° C., or 140° F. From this it follows that in preparing the calcium-solution most of the calcium hypasulphite formed is again decomposed, and that in its place calcium sulphate is obtained. Thus, a portion of the sulphur is completely wasted, which, with sodium sulphide, appears as hyposulphite. The calcium sulphide solution can only contain a large amount of the hyposulphite salt after it has been exposed to the oxidizing influence of the air. Both the sodium and calcium sulphide solutions, if exposed to the air, oxidize with formation of hyposulphite salts, but the latter solution does so more rapidly than the former.

Mr. Ottokar Hofmann, who used calcium sulphide as precipitant, at the Silver King Mill, Arizona, stated to me that the original sodium hyposulphite solution was used over a year and a half, and that it increased in strength and volume, making it necessary to run a part of it to waste. In this case a large amount of copper and lead was precipitated with the silver.

At La Dura, Sonora, Mexico, the lixiviation-solution was allowed to flow into the river by the stupidity of a laborer. A new supply of sodium hyposulphite could not have been obtained in less than ninety days, and in this dilemma, Mr. Hofmann proceeded to manufacture a new solution from the calcium hyposulphite con-

tained in the sulphide. The first wash-water, containing the base metals, was precipitated with calcium sulphide, and the operation repeated until a calcium hyposulphite solution of sufficient strength had been obtained to resume operations.

While these facts show that a calcium sulphide solution, unless freshly prepared, is by no means deficient in hyposulphite, they also prove that such a solution oxidizes too rapidly to be economical in practice.

§ 603. *Economy of producing Hyposulphites by Oxidation of Sulphide Solutions.*

The question is of practical importance, whether it is cheaper to buy and add sodium hyposulphite to a deteriorated lixiviation-solution, or to introduce hyposulphite by allowing the sulphide solution to oxidize, whereby it degenerates in precipitating power, but gains in hyposulphite. This depends altogether upon the price of chemicals, including freight. A calculation, under assumed conditions, will be of interest. For sodium sulphide, Na_2S_2, we find: To form 100 pounds $Na_2S_2O_3+5aq.$, from Na_2S_2, 37 pounds commercial caustic soda of 87 per cent. NaHO, and 25.8 pounds sulphur will be consumed.

37	pounds caustic soda, at 3 cts.	$1.11
25.8	" sulphur, at 1¼ cts.	32
62.8	" freight, at 2 cts.	1.26
	Total	$2.69

100 pounds sodium hyposulphite at 2 cts., and freight at 2 cts., cost $4.00, showing a profit of $1.31, if this salt is obtained by oxidation of Na_2S_2 in the mill. To go to the other extreme, allowing the sulphide solution to oxidize to such an extent that, finally, lixiviation-solution runs to waste, according to Mr. Hofmann's practice at Silver King, is not economical either.

§ 604. *Circumstances influencing the Value of the Precipitating-Coefficients of Sodium and Calcium Sulphide.*

The chemistry of the calcium and sodium polysulphides is extremely complicated, as demonstrated by Mr. Russell's numerous experiments, and the author's own observations. The precipitating-

coefficients of these solutions, namely, the quantities of silver precipitated for 100 parts of caustic soda consumed in the manufacture of sodium sulphide, or for 100 parts of sulphur consumed in the manufacture of either sodium or calcium sulphide, are so variable, that the slightest change of conditions produces different results. The most important elements of which these precipitating-coefficients are functions, may be enumerated as follows, viz:

1st. The mode of preparation.—This applies, especially, to sodium sulphide, which permits of greater variations in its manufacture than calcium sulphide. Concentration of the lye, temperature, etc., exert their influence.

2d. The proportions between sulphur and caustic soda, or caustic lime.—Here, as already pointed out, calcium sulphides of the formula CaS_4 and CaS_5 only can be obtained. Commercial caustic soda of good quality generally contains 87 per cent. NaHO. For the formation of different sodium polysulphides, the quantities of sulphur consumed would be as follows, viz:

100 caustic soda, of 100 per cent. NaHO, require:		100 caustic soda, of 87 per cent. NaHO, require:	sulphur to form
53.3	or	46.4	" Na_2S
80.0	"	69.6	" Na_2S_2
106.6	"	92.8	" Na_2S_3
133.3	"	116.0	" Na_2S_4
160.0	"	139.2	" Na_2S_5

3d. The concentration of the sulphide solutions.—Dilution has a tendency to diminish the precipitating-coefficients materially. The highest coefficients are only obtained from very concentrated sodium sulphide solutions. Calcium sulphide solutions are comparatively very diluted; hence, their precipitating power can never reach that of the sodium compounds.

4th. The proportion of silver to sodium hyposulphite in a lixiviation-solution.

5th. The concentration of the latter both in silver and sodium hyposulphite.

6th. Whether the silver is only partially or completely precipitated.

Mr. Russell's object in making these experiments was to find a method by which sodium sulphide solutions could be manufactured quickly, and with a minimum consumption of sulphur, giving high precipitating-coefficients. In this he has succeeded admirably. If sodium sulphide solutions are prepared according

to Russell's directions, as given in § 1205, consuming 66 parts sulphur for 100 parts commercial caustic soda, they are composed of both Na_2S and Na_2S_2, principally the latter.

These solutions show remarkably high precipitating-coefficients, namely, from 278 to 348 parts of silver precipitated, for 100 parts of sulphur consumed; or from 183 to 230 parts of silver precipitated, for 100 parts of caustic soda consumed. In the preparation of calcium polysulphide, the best coefficients were from 98 to 132 parts of silver precipitated for 100 parts of sulphur consumed, and that with freshly made solutions. Considering how much the latter deteriorate, after some time, it is safe to assume that the precipitation of silver by calcium polysulphide requires about three times as much sulphur as is needed with properly prepared sodium sulphide.

§ 605. *Theory of high Precipitating-Coefficients.*

Most interesting is the fact that the coefficients for caustic soda are much higher than the one expected according to the theory that, if a heavy metal is precipitated by an alkaline polysulphide, RSx, one equivalent of the latter precipitates not more than one equivalent, or a double equivalent of the former, $(x-1)S$, being liberated as free sulphur. If this theory was always true, and taking into consideration the reaction:

$$6NaHO + 2(x+1)S = Na_2S_2O_3 + 2Na_2Sx + 3H_2O.$$

100 parts caustic soda, with 87 per cent. NaHO, consumed in the manufacture of sodium sulphide, could not precipitate more than 156.6 parts silver, while the value for pure caustic soda would be 180. These phenomena can be explained by reactions similar to those taking place in the decomposition of extra-solution at ordinary and elevated temperature. At ordinary temperature, I noted the formation of tetrathionate salts; at elevated temperature, the formation of free sulphuric acid, besides Cu_2S. It has also been stated that the latter decomposition occurs, occasionally, at ordinary temperature.

In case tetrathionate salts were formed, the reaction would be expressed by the formula:

$$\text{I.}\quad 2Ag_2S_2O_3 + Na_2S_2 = 2Ag_2S + Na_2S_4O_6.$$

The theoretical precipitating-coefficient for pure caustic soda would be =360; or for the commercial article of 87 per cent. =313. The loss in sodium hyposulphite would be one equivalent for two equivalents of silver precipitated. Free sulphur would be entirely absent in the precipitate. This does not agree with the facts. It is neither possible to obtain a solution of Na_2S_2 with such high precipitating-coefficient, nor to precipitate Ag_2S free from sulphur by a sodium polysulphide. If we assume that free sulphuric acid is formed, the reactions would be as follows, viz:

$$2Ag_2S_2O_3 + Na_2S_2 + H_2O = 2Ag_2S + Na_2S_2O_3 + H_2SO_4 + S.$$

The sulphuric acid "in statu nascendi" would decompose Na_2S_2:

$$Na_2S_2 + H_2SO_4 = Na_2SO_4 + H_2S + S.$$

The sulphuretted hydrogen "in statu nascendi" would precipitate silver:

$$Ag_2S_2O_3 + H_2S = Ag_2S + SO_2 + S + H_2O.$$

The final reaction would be as follows, viz:

$$\text{II.}\quad 3Ag_2S_2O_3 + 2Na_2S_2 = 3Ag_2S + Na_2S_2O_3 + Na_2SO_4 + SO_2 + 3S.$$

The theoretical precipitating-coefficient for pure caustic soda would be =270; for the commercial article of 87 per cent. =235. (The exact figure is 234.9.) For three equivalents silver precipitated, one equivalent sodium hyposulphite would be lost, and for two equivalents silver precipitated one equivalent free sulphur would appear. It is remarkable how close the theoretical precipitating-coefficient for commercial caustic soda agrees with Russell's actual maximum value of 230, especially if we consider that his percentage of sulphur is a little short of the amount required to convert the caustic soda into Na_2S_2. This theory receives further support from the fact that free sulphur is actually precipitated.

An obscure critic has taken me to task for advancing this theory in my first paper on the Russell process, because, he says, not a trace of sulphuretted hydrogen is evolved in the reaction. Evidently, his classical education is at fault in not comprehending the meaning of "in statu nascendi."

The sulphuretted hydrogen evolved in precipitating sulphides on a large scale is principally due to the presence of acid salts in the lixiviation-solution, and can not be produced by the reactions of formula II., except when over-precipitation has taken place. (See

§ 1603.) The loss in sodium hyposulphite is, in part, made good by the addition of this salt present in the sodium sulphide. On a purely theoretical basis, the loss would be reduced one-half in formula II. For 100 ounces silver precipitated, the loss would be $2\frac{62}{100}$ pounds of $Na_2S_2O_3+5aq$. For one pound of silver it would be 0.382 pounds of $Na_2S_2O_3+5aq$. These figures, it should be remembered, refer to the actual loss in sodium hyposulphite to which the regenerated lixiviation-solution would be subjected, the lost sodium hyposulphite derived from the sodium sulphide not being counted at all.

Finally, another reaction should not be overlooked, namely, the conversion of Na_2S_2 by SO_2 into $Na_2S_2O_3$—see formula II.—according to the reaction:

$$2Na_2S_2+3SO_2=2Na_2S_2O_3+3S.$$

If sulphurous acid, "in statu nascendi," actually produced this effect, the end-reaction would be as follows, viz:

$$\text{III. } 9Ag_2S_2O_3+8Na_2S_2=9Ag_2S+5Na_2S_2O_3+3Na_2SO_4+12S.$$

A calculation, however, shows that reaction III. would produce a maximum silver precipitating-coefficient of only 176. This being in contradiction to facts, formula III. does not deserve further consideration.

§ 606. *Economy of using Sodium Sulphide with hiyh Precipitating-Coefficient.*

The question should be considered, whether, on account of the heavy loss in sodium hyposulphite, it is profitable or not to use a sodium sulphide of maximum precipitating-coefficient. The following calculations will show this for reaction II.

For precipitation of 100 pounds silver, by Na_2S_2 of highest precipitating-coefficient, =235, there are consumed and lost:

42.55	pounds commercial caustic soda, at 3 cts.	$1.28
29.61	" sulphur, at 1¼ cts.	.37
38.20	" sodium hyposulphite, at 2 cts.	.76
110.36	" freight, at 2 cts.	2.21
	Total	$4.62

For precipitation of 100 pounds silver, by Na_2S_2 of low precipitating-coefficient, =156.6, according to old formula, without loss in sodium hyposulphite, there are consumed and gained:

63.85	pounds commercial caustic soda, at 3 cts.	$1.81
44.44	" sulphur, at 1¼ cts.	.55
108.29	" freight, at 2 cts.	2.17
	Total	$4.63
	Gain in sodium hyposulphite, 57.4 pounds, at 4 cts., including freight	2.30
	Actual cost	$2.33

This would give an increased expense of $2.29 per 100 pounds silver precipitated by Na_2S_2 of highest coefficient, as compared with low coefficient, assuming that in the latter case sodium hyposulphite is not decomposed.

For 100 ounces silver precipitated, the increased expense would be about 16 cents.

If my theory is correct, the use of Na_2S_2 with highest precipitating-coefficient is not economical. Neither is it profitable to employ a reagent of low precipitating-coefficient, if the solution gains strength in sodium hyposulphite to such a degree that it becomes necessary to run a part of it to waste. The latter condition can not take place if extra-solution is used in lixiviation, because the manufacture of extra-solution causes a considerable loss in sodium hyposulphite.

These arguments should not be used against sodium sulphide generally. Mr. Russell found that the precipitating-coefficients of calcium sulphide solutions were just as variable. Hence, this reagent also must produce, under certain conditions, the same reactions as those pointed out for sodium sulphide.

§ 607. *Sodium Mono-sulphide.—Precautions in preparing Sodium Sulphide.*

The question may be asked: If a Na_2S_2 solution of low precipitating-coefficient is most profitable, why not use Na_2S, and save sulphur? The objection to the mono-sulphide is, that, in contact with air, it decomposes with formation of caustic soda:

$$2Na_2S + H_2O + 4O = Na_2S_2O_3 + 2NaHO,$$

while in the decomposition of Na_2S_2, sodium hyposulphite only is formed:

$$Na_2S_2 + 3O = Na_2S_2O_3.$$

Besides, a solution with only sufficient sulphur to form monosulphide, is liable to contain caustic soda and polysulphides. In consequence, free caustic soda would be largely introduced into the lixiviation-solution. It should be remembered, that caustic soda precipitates from a hyposulphite solution neither silver nor copper, but only lead. In fact, such a solution may have a precipitating-coefficient for caustic soda far below the theoretical value of 156.6. This may even occur if caustic soda is boiled with sufficient sulphur to form Na_2S_3. I will illustrate this by selecting a few examples from Mr. Russell's numerous experiments:

Solutions prepared by heating 1000 c. c. water with:		Silver precipitated per 100 gm. caustic soda:
400 gm. caustic soda.	400 gm. sulphur.	95 gm.
150 " "	150 " "	108 "
100 " "	100 " "	151 "

This can only be explained by assuming that sodium combines with sulphur to higher polysulphides, Na_2S_4 and Na_2S_5, leaving a portion of the caustic soda free. Hence, it would be risky, in manufacturing Na_2S_2, to aim at the production of a reagent with low precipitating-coefficient. If, for instance, a concentrated lye of pure caustic soda is heated in a water-bath with sufficient sulphur to form Na_2S_2, the precipitating-coefficient of the solution is far below the theoretical value of 180. It is absolutely necessary that the temperature of the lye should rise considerably above the melting-point of sulphur in order to form Na_2S_2, and not higher polysulphides, leaving caustic soda free.

§ 608. *Influence of diluting a Sodium Sulphide Solution upon the Value of its Precipitating-Coefficient.*

It is essential that the dilution of a sodium sulphide solution should not be carried beyond a certain point, or else its precipitating-coefficient is at once lowered. Here, again, a molecular change evidently takes place, with formation of higher polysulphides. The highest coefficients are obtained, if the solution is so concentrated that it measures 300 c. c. for each 100 gm. caustic soda consumed in its manufacture.

§ 609. *Decomposition of Sodium Hyposulphite by Heating.*

Another reaction may come into play in the preparation of sodium sulphide according to Russell's directions, having a tendency to increase the precipitating-coefficient, also at the expense of sodium hyposulphite. I refer to the decomposition of this salt, if heated above 100° C., or 212° F. In manufacturing sodium sulphide, the temperature of the mass in the tank rises to about 145° C., or 293° F., certainly not above 160° C., or 320° F.

1 gr. $Na_2S_2O_3+5aq.$ was placed in a test-tube, and immersed for half an hour in a paraffine-bath, heated to 160° C., or 320° F. The result was, that the salt remained perfectly white, and did not decompose at all. A test with $BaCl_2$ did not show a trace of sulphate. The decomposition into sulphur, sodium sulphide and sulphate, requires a very much higher temperature. This proves conclusively that there is no danger of decomposing sodium hyposulphite in the manufacture of the sulphide.

§ 610. *Precipitating-Coefficients of Sodium Sulphide for Lead and Copper.*

It may be assumed, on purely theoretical grounds, that, in precipitating lead and copper from hyposulphite solutions, the same reactions will occur as in the case of silver. Copper, being present as a cuprous salt, is precipitated as Cu_2S. In order to obtain the precipitating-coefficients for lead and copper, the value for silver should be multiplied by 0.958 for the former, and by 0.588 for the latter metal. The loss and gain in sodium hyposulphite will be in proportion to the quantities of the reagent consumed. For lead, the results are so nearly the same as those for silver, that, for practical purposes, they may be considered equal. A marked difference, however, exists for copper, so that a calculation as to loss and gain in sodium hyposulphite will be of interest.

The low precipitating-coefficient of Na_2S_2 for copper would be $=92$; the highest $=138.2$.

For precipitation of 100 pounds copper, by Na_2S_2 of highest precipitating-coefficient $=138.2$, there would be consumed and lost:

formula, without decomposition of sodium hyposulphite. It depends altogether upon local circumstances, whether it is more profitable to use Na_2S_2 of high or low coefficient. By dilution of the solution to a proper degree, the most profitable effect can be produced.

It will be seen that this theory agrees perfectly with all the facts.

B. Calcium and Sodium Sulphide compared as Precipitants for Silver.

Kuestel and Hofmann claim that sodium sulphide is too expensive on account of the high price of caustic soda, especially in a country like Mexico, while caustic lime costs almost nothing. How ill founded this argument is the following statistics will demonstrate.

§ 613. *The Cost of Precipitating with Calcium and Sodium Sulphide at the Cusihuiriachic Mill.*

The results given below were obtained at the Cusihuiriachic Mill, Mexico. The mill, reducing 50 tons of ore per day, was run 38 days, using sodium sulphide, and 21 days, using calcium sulphide as precipitants for silver. The quantities of chemicals consumed were as follows, viz:

Mill running 38 days, using Sodium Sulphide:

Average value of ore, 35.1 ounces silver per ton.

Consumption of	caustic soda, per ton	4.4	pounds.
"	sulphur, "	2.9	"

Mill running 21 days, using Calcium Sulphide:

Average value of ore, 39.0 ounces silver per ton.

Consumption of	caustic lime, per ton	24	pounds.
"	sulphur, "	10.3	"

For better comparison we have to reduce the above figures to what they would be in working ore of 35.1 ounces, per ton, and in doing so find:

Consumption of	caustic lime, per ton	21.6	pounds.
"	sulphur "	9.3	"

Outside of a difference in value, the ore was exactly of the same character in both cases. At the Cusihuiriachic mill the prices for chemicals were: sulphur, 7 cents; caustic soda, 10 cents;

caustic lime, 1 cent per pound. Hence, the cost of sodium sulphide, per ton of 35-ounce ore, was 64.3 cents; and of calcium sulphide, 86.7 cents, or 22.4 cents in favor of the former, resulting in a saving of $11.20 on 50 tons of ore per day.

There is, however, another important point to be taken into consideration, namely, the labor, time, and steam consumed in manufacturing the two reagents. At Cusihuiriachic 4 iron tanks, 2½ feet by 3 feet by 6 feet, had to be kept boiling day and night, without interruption, in order to supply sufficient calcium sulphide for 50 tons of ore; while in 1½ hours enough sodium sulphide was obtained in one tank, by Russell's method, to last for three days, or for 150 tons of ore.

It can readily be seen that calcium sulphide will be less expensive only where sulphur is very much cheaper than caustic soda, and where caustic lime costs almost nothing. Even then, the preference should be given to sodium sulphide on account of its superiority in every respect.

§ 614. *Calcium and Sodium Sulphide compared as Precipitants, mechanically speaking.*

G. Kuestel states—and this has been copied by others—that in using calcium polysulphide, the sulphides precipitate quicker, and settle better than with a sodium polysulphide. This statement is not confirmed, but contradicted by Russell's experiments. In working on a large scale, the sulphides precipitated by sodium monosulphide, or polysulphide, settled without difficulty, and in a short time, and the effect of calcium sulphide did not show itself to be superior in any respect. On the contrary, sodium monosulphide or bisulphide, proved to be infinitely better than calcium pentasulphide. Indeed, it is not possible to advance theoretical reasons to support the superiority of calcium polysulphide. Using the pentasulphide, which is mostly obtained, much more sulphur is wasted, and appears as free sulphur in precipitating the lixiviation-solution.

§ 615. *Precipitation of Gypsum by Calcium Sulphide.*

Another point against the use of calcium sulphide is the fact that gypsum is precipitated with the sulphides, whereby their subsequent treatment becomes more troublesome.

Mr. Russell found that calcium sulphate is, practically, " insoluble in a solution of calcium sulphide." (And also in sodium sulphide.) Hence, no gypsum can enter the lixiviation-solution from this source directly. If, however, to a calcium sulphide solution a soluble sulphate is added, for instance that of sodium, a precipitate of gypsum appears immediately; and if to a sodium hyposulphite solution, containing sodium sulphate, or any other sulphate, calcium sulphide is added, gypsum is formed which, in part, remains in solution, and, in part, is thrown down with the sulphides.

I now turn to the question: In what respect differ the calcium and sodium hyposulphite solutions regarding their efficiency in lixiviation? The points to be considered are: The deterioration of the solutions after prolonged use; their dissolving energy for extracting silver; their dissolving energy for extracting gold, in case gold-bearing silver ores are lixiviated.

§ 616. *The Deterioration of a Lixiviation-Solution after Prolonged Use.*

If a hyposulphite solution is left to itself for a long time it deteriorates in a marked degree. It takes up oxygen from the air, and sulphates are formed. Mr. Russell interrupted on two occasions, at the Ontario mill, his lixiviation-experiments, on a large scale, for some time, and determined the deterioration of the sodium hyposulphite solution.

Solution not used for twenty-six days.

100 c.c. dissolved,	0.485 gm. silver as AgCl.
After twenty-six days,	0.458 gm. " "
Deterioration,	5.5 per cent.

Solution not used for thirty-five days.

100 c.c. dissolved,	0.447 gm. silver as AgCl.
After thirty-five days,	0.359 gm. " "
Deterioration,	17.4 per cent.

This result is interesting, because it shows that the decomposition of the solution is not merely a function of time. I am inclined to believe that, outside of difference in temperature, and surface exposure, the condition of the atmosphere has a marked influence, most likely its contents in ozone.

In order to test the relative deterioration of solutions of calcium and sodium hyposulphite, Mr. Russell has carried out the following experiments.

Solutions of different concentration were exposed in soup-plates to the atmosphere for seven days at a temperature of from 20° to 22° C., or 68° to 71° F., and brought to their original volume at the expiration of that time. The relation of the depth of the solution to the diameter of the plate was about 1 to 8. A re-determination of the solvent energy of these solutions for silver chloride gave the following average results, namely: The sodium hyposulphite solutions had deteriorated $1\frac{4}{10}$ per cent.; the calcium solutions had deteriorated $16\frac{1}{10}$ per cent.

§ 617. *The Solvent Energy of various Hyposulphite Solutions for Silver Chloride.*

Mr. Russell prepared solutions of $1\frac{1}{2}$ per cent. concentration from the hyposulphites of sodium, potassium, and calcium, and treated with them freshly precipitated silver chloride under equal conditions. I present the results of these experiments as follows, calling the solvent energy of sodium hyposulphite 100.

Name of salt.	Solvent energy for silver chloride.
Sodium hyposulphite	100.
Potassium "	70.
Calcium "	91.5

This shows plainly the inferior effect of the calcium-salt. That of potassium hyposulphite is comparatively low and does not seem to have been investigated before. It finds a parallel in the inferior solubility of silver chloride in potassium chloride solutions as compared with those of sodium chloride.

§ 618. *The Dissolving Energy of Sodium and Calcium Hyposulphite Solutions for Gold.*

We find it stated in treatises on metallurgy that calcium hyposulphite is a more energetic solvent for gold than the sodium-salt, and, in copying this statement, metallurgical writers have gone so far as to assert that sodium hyposulphite does not extract gold at all.

It seems to me that there is no scientific foundation for this statement, the double-salt of $3Na_2S_2O_3+Au_2S_2O_3+4aq.$ being as easily soluble as the calcium double-salt. To settle this question, I induced Mr. Russell to carry out some comparative experiments with ore from the Lexington, Montana, and from the Custer, Idaho. For the sake of completeness Mr. Russell also used solutions of potassium and magnesium hyposulphite. In all cases the solutions were of 2 per cent. concentration, and the figures below represent the averages of six tests.

	Ounces gold remaining in tailings after lixiviating with the hyposulphite of:			
Sample from:	Sodium.	Potassium.	Calcium.	Magnesium.
Lexington Mill, . .	0.333	0.316	0.326	0.366
Lexington Mill, . .	0.308	0.320	0.313	0.366
Custer Mill, . .	0.280	0.350	0.362	0.280

These figures prove conclusively that the superiority of a calcium hyposulphite solution for the extraction of gold from roasted silver-ores is a fable.

CHAPTER VII.

Laboratory Work.

In a lixiviation-mill the following determinations should be made every day:

1. Pulp-assay of the ore and salt mixture.
2. Determination of the percentage of salt in the pulp.
3. From these data the value of the ore is calculated.
4. Pulp-assay of the roasted ore.
5. Determination of the percentage of soluble salts in the roasted ore.
6. From these data the value of the roasted ore, minus soluble salts, is calculated.
7. Tailings assay.
8. In comparing the value of the tailings with the value of the roasted ore, minus soluble salts, the percentage of silver left in the tailings is calculated.
9. Lixiviation-tests of the roasted ore with ordinary solution.
10. Lixiviation-tests of the roasted ore with extra-solution.

The results of No. 9 show to what perfection the roasting has been carried; and those of No. 10 show what results may be expected in lixiviating on a large scale. By comparing No. 9 with No. 10 we learn how much the extraction of the silver will be benefited by the use of Russell's process.

If the ore is treated raw, most of these determinations are not required.

Silver in wash-water, in treating roasted ores, is determined by precipitating a measured quantity with sodium sulphide, collecting the precipitate on a filter, and assaying it for silver.

Of other work, there is the volumetric determination of sodium hyposulphite in the stock-solution, and testing ores on a small scale regarding their fitness for lixiviation.

§ 701. *Determination of the Percentage of Salt in Crushed Ore.*

To determine the percentage of salt, take 10 gm. of the pulp, put it on a filter, and leach with hot water until the filtrate ceases to give a reaction with silver nitrate. For drying of the residue, the funnels are placed in a drying-chamber, heated by steam or boiling water. The dry residue is either weighed with the filter, the weight of which has been previously ascertained, or it is removed from the filter with a brush.

§ 702. *Determination of Soluble Salts in Roasted Ore.*

The same "modus operandi" can be adopted as in § 701; subsequent treatment with sodium hyposulphite solution not being absolutely necessary. In most cases the result of this determination will agree sufficiently close, say within 1 per cent., with the percentage of soluble salts actually extracted in leaching and lixiviation on a large scale. If, however, the roasted ore contains considerable calcium sulphate and caustic lime, both of which are not easily soluble either in water or in sodium hyposulphite solution, the laboratory-test will indicate a much higher solubility than that reached in the mill, and a correction must be made. The reason for this discrepancy is, that in operating on a large scale, the calcium sulphate and the caustic lime are only partially removed, owing to the much smaller quantity of solution used in proportion to the ore than in the laboratory-test. The correction is made as follows, viz: 10 gr. of the roasted ore, and the same amount of the tailings corresponding to the ore, are placed on filters. Each is leached first with about 300 c. c. of cold water, and then lixiviated with a cold sodium hyposulphite solution of about 10 per cent. concentration, until the filtrate ceases to give a reaction with sodium carbonate, indicating that all the calcium-compounds are extracted. Water and sodium hyposulphite solution, if cold, it will be remembered, are better solvents for calcium hydrate and sulphate than if hot. The residues are washed with water, dried and weighed. The difference between the percentage of solubility of the ore, and that of the tailings, gives, with suffi-

cient exactness, the percentage of salts extracted in the mill by both water and lixiviation-solution. The slight error is due to the fact that the ore and tailings are not of the same nature—the latter having been concentrated previously to the weighing, by an amount equal to the solubility in the mill; but the determination is sufficiently exact for practical purposes.

§ 703. *Lixiviation-Test with Ordinary Solution.*

The quantity of ore used is ½ A. T., or less if desirable. Roasted ores are first leached with 300 c. c. water. After decanting the clear wash-water, 250 c. c. of a 5 per cent. sodium hyposulphite solution are added to the ore in the beaker, and the latter is heated on a sand-bath until a temperature of about 55° C., or 131° F., has been reached. Stirring the solution frequently, it is kept at that temperature for half an hour. Now, the solution is decanted on the same filter to which the tailings are transferred. The filter, after washing the tailings with water, is manipulated as in § 701, and the tailings assayed. The tedious operation of burning the filter is entirely unnecessary, the small amount of dust, that may be left on the paper, not being sufficient to influence the result in a perceptible degree.

The method of making the ordinary lixiviation-test, as described above, will suffice in most cases. Some ores, however, require a different treatment. This is especially the case with roasted arsenical ores, containing considerable lime. At Yedras, for instance, the ordinary lixiviation-test is not made in beakers, but the sample is treated on a filter with a boiling sodium hyposulphite solution, until the filtrate is free from silver. It is not necessary to point out other modifications, which can easily be devised to meet special cases.

§ 704. *Influence of re-pulverizing Roasted Ore in an Iron Mortar.*

In case roasted ores have been so coarsely crushed in the mill as to require a re-pulverizing for assaying, an iron mortar should not be used. As shown in Table No. 704, the extraction of silver from Cusi ore, by ordinary solution in the assay-office, is reduced

10 to 11 per cent., while results with extra-solution are practically unaltered. This is caused by reduction of silver to the metallic state, in which it is much more soluble in extra-solution than in ordinary solution.

On Lake Valley ores, the use of the iron mortar also reduced the extraction by extra-solution 18 per cent. I am not sufficiently familiar with the character of these ores to account for the difference.

TABLE No. 704.

EXPERIMENTS ON THE EFFECT OF GRINDING ROASTED ORE IN WEDGEWOOD OR AGATE, AND IN IRON MORTARS.

Ore from:	Mesh of Screen and Method of Crushing.	Sample from:	Material of Mortar.	Value of Ore.	Value of Tailings from Ordinary Solution.	Value of Tailings from Extra-Solution.	Per cent. Extracted by Ordinary Solution.	Per cent. by Extra-Solution.
				Ounces per ton.				
Cusi.	20-Screen. Stamps.	Furnace	Wedgewood.	30.2	6.4	2.8	78.9	90.7
		"	Iron.	30.2	9.6	2.8	68.3	90.7
		Lixiviation-Vat	Agate.	30.0	4.4	2.6	85.3	91.3
			Iron.	30.0	7.4	2.8	75.4	90.3
Lake Valley.	16-Screen. Rolls.	Furnace	Agate.	16.0	2.9	2.0	81.3	87.5
		"	Iron.	16.0	6.7	6.0	58.1	62.5
		Cooling-floor ...	Agate.	18.0	3.7	3.5	79.4	80.6
			Iron.	18.0	7.2	6.0	60.0	66.7
		Lixiviation-Vat	Agate.	18.0	3.0	3.0	83.3	83.3
			Iron.	18.0	9.0	5.7	50.0	68.3

NOTE.—The screen used in the assay-office was No. 40 wire-cloth.

§ 705. *Influence of Copper Vessel.*

It is customary, in some mills, to heat the sodium hyposulphite solution, used in the assay-office, in a copper vessel. Copper is dissolved, as shown in § 206, and a weak extra-solution formed. According to Mr. F. Johnson's observations, the effect of this extra-solution may be considerable, as shown in Table No. 705.

TABLE No. 705.

EFFECT ON THE LIXIVIATION-TESTS WITH ORDINARY SOLUTION, BY HEATING THE LATTER IN A COPPER VESSEL.

Roasted Ore.	Value in Ounces per Ton.		Ounces in Favor of Copper Vessel.	Per cent. in Favor of Copper Vessel.
	Tailings from Solution heated in Copper Vessel.	Tailings from Solution heated in Glass Vessel.		
73.80	20.52	24.84	4.32	5.8
74.76	8.88	10.32	1.44	2.0
77.46	10.92	15.24	4.32	5.6
60.00	13.80	20.16	6.36	10.4
72.00	15.30	19.56	4.26	5.9
75.60	9.84	11.52	1.68	2.2
76.32	9.84	12.90	3.06	4.1
63.96	14.94	16.82	1.88	3.0
69.30	15.96	18.60	2.64	3.8
72.78	8.40	9.12	1.02	5.4
72.72	8.28	10.98	2.11	3.7
71.70	12.42	15.46	3.0	4.7

LIXIVIATION-TEST WITH EXTRA-SOLUTION.

In my second paper on the Russell process, read before the Am. Inst. of M. E., at the St. Louis meeting, in October, 1886, I stated that Mr. Russell used the following three methods in making lixiviation-tests with extra-solution, viz:

A. The sample is treated with extra-solution only.

B. The sample is treated as follows: Add 20 c. c. copper solution and heat to 100° or 125° F.; leave for five minutes, and now add 300 c. c. hyposulphite solution, and heat to 125° or 130° F. Decant upon filter, wash out, etc.

C. The sample is treated with ordinary solution; decanted and treated with extra-solution.

For raw ores method C is most frequently used. A often gives as good results. B is rarely used.

Roasted ores are generally first leached with water. If not much salt has been used in roasting, and the sample contains only a small amount of soluble base-metal salts, this operation may be dispensed with.

For roasted ores without caustic lime method C is most frequently used, but A may give as good results.

If the roasted ore contains caustic lime, method B alone is used.

An extended experience with a great variety of ores has demonstrated that these three methods are entirely insufficient to determine what percentage of the silver may be extracted on a large scale, and whether an ore is suitable for lixiviation or not. In addition to this, Mr. Russell found that in many cases identical methods, applied in the laboratory and in the mill, do not give equal results, but that there is in every case some one laboratory-test the results of which closely correspond to those of the most advantageous method used in the mill. This may appear contradictory at first sight; we must consider, however, that conditions in the laboratory can never be the same as those in working on a large scale. The methods of applying the extra-solution are, of course, variable without limit, and, to a great extent, arbitrary. In order to make the work in the laboratory of practical value, a limited number of tests must be selected that will cover all cases. Mr. Russell tested, by 22 different methods, samples of roasted ore, taken from 17 mills, which were selected with a view of representing great variety in character. He found that of these 22 methods, 14 could be dispensed with, while of the remaining 8, if applied to different ores, one or several gave maximum results. In the same way, 8 methods were selected for lixiviation-tests of raw ores.

§ 706. *Russell's Selection of Lixiviation-Tests with Extra-Solution, Criticised.*

In reviewing Mr. Russell's table recording his experiments, I find that the number of lixiviation-tests for roasted ores can be reduced to 5, and for raw ores also to 5. Concerning raw ores, practical experience is exceedingly limited, in fact, almost wholly lacking; hence, this subject must be considered incomplete. As to roasted ores, the three tests I propose to strike out are, Nos. II., VI., and VII. Method No. II. is almost identically the same as No. IV., the difference being that in No. II. leaching with water is omitted. These two methods, Nos. II. and IV., gave the best results with Yedras ore, namely, 90.0 and 90.5 per cent. Methods Nos. I., III., and VIII., gave from 80.5 to 85.6 per cent.; while methods VI. and VII. extracted only 37.0 per cent. To methods VI. and VII. I object on general principles. Solutions of such

abnormal concentrations are used, that I doubt whether it will be possible to obtain equal results in working on a large scale, and there is no proof furnished that my criticism is incorrect. The only sample requiring such abnormal treatment, in order to obtain maximum results, came from the North Mexican mill, Cusihuiriachic, Mexico. Methods Nos. VI. and VII. gave 82.0 and 82.9 per cent.; while Nos. I. and IV. extracted 78.7 per cent., No. V. 79.5 per cent., and other methods less.

§ 707. *Control of the Mill-Work by Laboratory-Tests essential in Lixiviation.*

To a novice in lixiviation, this laboratory-work looks much more formidable than it is in reality. It has to be considered that in working ore of a known character the severe work of many tests is only required in starting the mill. After the laboratory-test—corresponding most nearly to the best treatment on a large scale—has once been selected, the matter becomes simple enough. A close control of the mill-work, by tests in the laboratory, is much more essential in lixiviation than in treating ores by amalgamation, since the latter process, especially with roasted ores, can only be subjected to a few changes as compared with the former. In making the so-called chlorination-tests of roasted ore in a pan-mill we obtain a control of the work performed by the roasting furnaces, and nothing else. In many cases there is a close correspondence between the chlorination-test and the extraction by amalgamation; in other cases no such agreement exists. An ore may show a very high chlorination, and give comparatively low results in amalgamation. This is often due to the presence of silver antimonate and arsenate, which are, as we have seen, extracted by ordinary hyposulphite solution, but not amenable to amalgamation. From such tailings a large percentage of the silver can be extracted by lixiviation. It also occurs frequently that amalgamation shows a higher yield than the chlorination-test, and even then, silver remains in the tailings that can be extracted by lixiviation. In this instance, the difference may be due to metallic silver, and to other causes not known.

§ 708. *General Remarks regarding the Lixiviation-Tests with Extra-Solution.*

In the following lixiviation-tests the quantity of ore used is ½ A. T., or less if desirable. The copper solution is made by dissolving 200 gm., $CuSO_4+5aq.$ in water, and diluting to 1000 c. c. The treatment of the samples is done in beakers. If beakers of less capacity than 300 c. c. are on hand, the volume of the solution can be reduced, but the quantity of the chemicals remains the same. The contents of the beakers should be occasionally stirred. The preliminary leaching with about 300 c. c. water is done by decantation of the clear part of the solution upon the same filter to which the tailings are finally transferred. The tailings on the filter are treated in the same way as in the ordinary lixiviation-test. For better oversight of these lixiviation-tests I have grouped them somewhat differently, as has been done in Mr. Daggett's paper, retaining, however, Mr. Russell's original numbers. The numbers I propose to abolish are enclosed in parenthesis. For the sake of brevity, the terms Hypo. and Copper are used for sodium hyposulphite and for copper sulphate solution.

§ 709. *Lixiviation-Tests with Extra-Solution for Roasted Ores.*

Methods in which leaching with water is omitted; Copper is added first.

A. The solutions are not heated.

a. The time of exposure is short.

I. Add 50 c. c. cold water and 25 c. c. Copper; let stand 2 hours cold.

Dilute to 300 c. c. with cold water; add 20 gm. Hypo.; let stand 2 hours cold; filter and wash.

III. Add 250 c. c. cold water and 25 c. c. Copper; let stand 2 hours cold.

Add 20 gm. Hypo.; let stand 2 hours cold; filter and wash.

b. The time of exposure is long.

[II.] Add 250 c. c. cold water and 25 c. c. Copper; let stand 12 to 18 hours cold.

Add 20 gm. Hypo.; let stand 12 to 18 hours cold; filter and wash.

B. The solutions are heated; the time of exposure is short.

V. Add 250 c. c. cold water and 25 c. c. Copper; let stand 1 hour cold.

Add 20 gm. Hypo.; heat to about 55° C., or 131° F.; decant.

Add 250 c. c. cold water and 25 c. c. Copper; let stand 1 hour cold.

Add 20 gm. Hypo.; heat to about 55° C., or 131° F.; filter and wash.

Methods in which the sample is first leached with water.

A. The solutions are used cold; Copper is added first; the time of exposure is long.

IV. Leach with cold water.

Add 25 c. c. Copper; dilute to 300 c. c. with cold water; let stand 12 to 18 hours cold.

Add 20 gm. Hypo.; let stand 12 to 18 hours cold; filter and wash on filter with hot Hypo. solution.

B. The solutions are heated; the time of exposure is short.

a. Copper is added first.

[VI.] Leach with hot water.

Add 200 c. c. hot water and 90 c. c. Copper; let stand one-half hour.

Add 90 gm. Hypo.; heat to about 65° C., or 149° F.; filter and wash.

[VII.] Leach with hot water.

Add 90 c. c. Copper; let stand one-half hour.

Add 90 gm. Hypo.; dilute to 300 c. c.; heat to about 65° C., or 149° F.; filter and wash.

b. Hypo. is added first.

VIII. Leach with hot water.

Add 20 gm. Hypo.; let stand (with residue of wash-water, about 30 to 40 c. c.) 1 hour.

Add 25 c. c. Copper; dilute to 300 c. c. with cold water; heat to about 55° C., or 131° F.; filter and wash.

The annexed Tables, Nos. 709. a, and 709. b, explain themselves:

TABLE No. 709. a.

METHODS USED, AND RESULTS OF LIXIVIATION-TESTS BY EXTRA-SOLUTION, IN ASSAY-OFFICE, ON ORES ROASTED IN STETEFELDT, HOWELL, REVERBERATORY AND BRÜCKNER FURNACES.

Name of Mine.	How Crushed.	Mesh of Screen.	Per cent. of Salt Used.	Kind of Furnace.	Per cent. of Silver Extracted.	Lixiviation-Tests Most Applicable.
Ontario, Park City	Stamps.	16	12	Stetefeldt.	97.1	VI. or VII.
Daly, " "	"	26	11	"	95.2	II. or I.
Lexington, Butte	"	30	15	"	93.2	VIII. or V.
Manhattan	"	40	6	"	96.7	I. or III.
Alice, Butte	"	26	15	Howell.	93.5	VII. or I.
Sierra Grande, Lake Valley	Rolls.	16	7	"	84.6	V. or VII.
Pyramid	Stamps.	30	------	"	93.6	VII. or VI.
San Antonio, Chihuahua	"	12	10	"	91.0	I. or VIII.
San Miguel, Chihuahua	"	12	8	"	93.6	I. or III.
Chloride, New Mexico	Rolls.	30	7	"	91.6	
Yedras, Sinaloa	Stamps.	26	7	Reverb'r'y.	90.5	IV. or II.
Sombrerete, Zacatecas	Rolls.	20	10	"	94.9	V. or III.
Carmen, "	"	20	0	"	86.7	III. or IV.
Veta Grande, Chihuahua	Stamps.	16	6	Brückner.	87.8	III. or I.
Custer, Idaho	"	30	5	"	95.1	I. or II.
Bertrand, Nevada	Rolls.	12	6	"	83.9	V. or VIII.
Tybo, Nevada	Stamps.	30	------	"	93.1	V. or I.
North Mexican, Chihuahua	"	30	------	"	82.9	VI. or VII.

Per ct. 86.5 — Per ct. 90.7 — Per cent. 91.3 — Per ct. 96.5

TABLE No. 709. b.

COMPARISON BETWEEN THE RESULTS BY RUSSELL PROCESS IN THE ASSAY-OFFICE AND IN THE MILL.

Name of Mine.	Kind of Furnace Used.	Mesh of Screen.	Per cent. of Salt.	Number of Assay-Office Extra-Solution Tests Tried.	Highest Result obtain'd in the Assay-Office, per cent.	Mill per cent.	Per cent. Extracted in Assay-Office, more or less than in Mill.
Ontario	Stetefeldt.	30	0	2	85.8	85.7	+ 0.1
"	"	30	9½	2	94.7	96.2	− 1.5
"	"	20	12½	2	97.0	97.0	0.0
"	"	16	7½	2	89.0	89.5	− 0.5
"	"	16	8	2	91.5	92.0	− 0.5
"	"	16	12	2	96.3	97.1	− 0.8
"	"	16	16	2	95.0	95.7	− 0.7
"	"	16	18	2	95.9	95.1	+ 0.8
Lake Valley	Howell.	16	7	2	82.6	85.3	− 2.7
Sombrerete	Reverb'y	8 screen but with fine half out.	10	2	86.2	86.5	− 0.3
"	"	8	10		90.6	91.3	− 0.7
San Bartolo	Howell.	16	9	8	88.5	87.3	+ 1.2
San Antonio	Howell.	26	10	8	89.2	88.5	+ 0.7
"	"	12	10	8	90.3	83.9	+ 6.4
San Miguel	Howell.	26	8	8	91.9	90.4	+ 1.5
"	"	12	8	8	89.1	84.0	+ 5.1
Yedras	Brückner.	26	7	8	83.1	82.6	+ 0.5

§ 710. *Lixiviation-Tests with Extra-Solution for Raw Ores.*

A. *Methods in which the solutions are used cold, and the time of exposure is long.*

[I.] Add 250 c. c. cold water and 20 gm. Hypo.: let stand 12 to 16 hours cold.

Decant; add 25 c. c. Copper; dilute to 300 c. c. with cold water; let stand 12 to 16 hours cold.

Add 20 gm. Hypo.; let stand 12 to 16 hours cold; filter and wash.

[II.] Add 250 c. c. cold water and 20 gm. Hypo.; let stand 12 to 16 hours cold.

Add 20 gm. Hypo. and 25 c. c. Copper; let stand 12 to 16 hours cold; filter and wash.

[III.] Add 100 c. c. cold water and 25 c. c. Copper; let stand 12 to 16 hours cold.

Dilute to 300 c. c. with cold water; add 20 gm. Hypo.; let stand 12 to 16 hours cold; filter and wash on filter with hot Hypo. solution.

B. *Methods in which the solutions are heated, and the time of exposure is short.*

IV. Add 100 c. c. cold water and 25 c. c. Copper; let stand 1 hour cold.

Add 200 c. c. cold water and 20 gm. Hypo.; heat to about 55° C., or 131° F.; filter and wash.

V. Add 250 c. c. cold water and 25 c. c. Copper; let stand 1 hour cold.

Add 20 gm. Hypo.; heat to about 55° C., or 131° F.; decant.

Add 100 c. c. cold water and 25 c. c. Copper; let stand 1 hour cold.

Add 20 gm. Hypo.; heat, filter and wash.

VI. Add 200 c. c. cold water and 90 c. c. Copper; let stand one-half hour cold.

Add 90 gm. Hypo.; heat to about 65° C., or 149° F.; filter and wash.

VII. Add 250 c. c. cold water and 25 c. c. Copper; heat to about 55° C., or 131° F.; decant.

Add 20 gm. Hypo. and 25 c. c. Copper; dilute to 300 c. c. with cold water; heat to about 55° C., or 131° F.; filter and wash.

VIII. Add 200 c. c. hot water and 90 c. c. Copper; let stand one-half hour.

Add 90 gm. Hypo.; heat to about 65° C., or 149° F.; filter and wash.

§ 711. *Volumetric Determination of Sodium Hyposulphite with Iodine.*

The method is based upon the reaction:

$$2\ Na_2S_2O_3 + 2\ I = 2\ Na\ I + Na_2S_4O_6,$$

and the property of free iodine to turn starch blue. Hence, if to a hyposulphite solution, containing starch, iodine is added, the blue color of the starch will appear as soon as all the hyposulphite has been changed to tetrathionate.

If c. p. iodine cannot be obtained, the commercial article should be purified. This is done by mixing the iodine with about $\frac{1}{6}$ of its weight, potassium iodide, and drying the mixture in a large watch-glass, with ground rim, by heating it slowly on a sand-bath. As soon as violet vapors appear the watch-glass is covered by a second one of equal diameter. Heating is continued until the iodine has sublimated. Chlorine and bromine, frequently found in common iodine, remain combined with potassium in the residue of potassium iodide.

The potassium iodide, used for dissolving the iodine, should be pure, and not contain free iodine. Its solution, made acid with c. p. hydrochloric acid, should not turn blue upon addition of starch solution.

Preparation of the Iodine Solution.—5.121 gm. iodine are placed in a litre-bottle with glass-stopper, together with 7.5 gm. potassium iodide, and 100 c. c. distilled water. A slight excess above this weight may be taken, as it is easier to correct the titre of a solution that is too strong than of one that is too weak. The iodine and potassium iodide are brought together in one spot, and the bottle is left in a dark and moderately warm place. If, after twenty-four hours, the iodine has not all dissolved, the solution is shaken, more potassium iodide is added, and the iodine is again collected in one spot so that it may come in contact with the potassium iodide crystals. After complete solution of the iodine has taken place, the litre-bottle is filled to the mark with distilled

water and shaken. The solution is kept in bottles with glass-stoppers, and in the dark. Under these circumstances it will be quite permanent. 1 c. c. of this solution should correspond to 0.010 gm. of sodium hyposulphite.

Preparation of the Starch Solution.—To one part of starch, mixed with a little water, 100 parts of boiling water are added. The solution is filled into a tall beaker, and allowed to settle. After decanting the clear solution the rest is filtered. Pure common salt is now added until saturation has taken place, and after the solution has become clear it is preserved in bottles of 3 or 4 ounces capacity. This solution will keep longer than one year. In place of salt, Mr. F. M. Watson adds a slight quantity of salicylic acid to the starch solution. The latter method is better and more convenient than the former.

Determination of the Titre of the Iodine Solution.—0.100 gm. c. p. sodium hyposulphite is dissolved in 20 c. c. water, to which 1 c. c. starch solution is added. Then the iodine solution is added from a graduated burette, until the starch turns blue. The burette should have a glass-cock. If the titre of the iodine solution is correct, 10 c. c. should have been consumed. If there is a perceptible difference, the titre of the solution must either be corrected, or the difference should be taken into account in testing the stock-solution.

Determination of the Concentration of the Stock-Solution in Hyposulphite.—Take, for instance, 10 c. c. of stock-solution, dilute to about 50 c. c., and proceed as above. If the titre of the iodine solution is correct, each 10 c. c. consumed represent 1 per cent. of sodium hyposulphite.

The stock-solution, before testing, should not be caustic, but either neutral or slightly acid. In case it is caustic, it should be neutralized with dilute sulphuric or hydrochloric acid. An excess of acid, however, must be avoided so as not to decompose hyposulphite salts.

§ 712. *Influence of Silver, Copper, and Lead on the Correctness of Results.*

It is important to know the effect of silver, copper, and lead present in a hyposulphite solution, if the latter is tested with iodine. In the following experiments, 0.200 gm. $Na_2S_2O_3+5aq.$

were in each case dissolved in water, and then salts of silver, lead, and copper were added in such quantities that their contents in these metals were 0.005 gm. To the copper solution, an extra amount of sodium hyposulphite was added so as to compensate for decomposition of this salt by copper sulphate, leaving exactly 0.200 gm. unchanged.

The silver solution became slightly turbid at the end of the reaction, but the latter was quick and definite, recording exactly 0.200 gm. $Na_2S_2O_3+5aq$.

The lead solution remained clear, the result being as accurate as above. If a large quantity of lead is present, the latter falls out as yellow iodide, the color of which interferes with observing the end of the reaction. Hence, if hyposulphite is to be determined in such a solution, the lead should first be separated by sodium carbonate.

The copper solution became very turbid, and the end-reaction was slow and indistinct. The sodium hyposulphite found was 0.225 gm. This is due to the presence of copper in the form of a cuprous salt. If hyposulphite is to be determined in a solution containing copper, the following method may be used.

Take equal parts of the solution, and find in one hyposulphite plus cuprous oxide; in the other the amount of copper. The latter can easily be separated as Cu_2S by boiling the acid solution. For practical purposes it is sufficiently accurate to deduct for each 0.005 gm. copper, 2 c. c. iodine solution, if the latter is of correct standard. If a large quantity of copper is present, it should first be precipitated as sulphide.

§ 713. *Assaying of the Sulphides for Silver.*

The set of experiments shown in Table No. 713 was made by Louis Janin, Jr., when in charge of the mill at Cusi. In experiments 1 and 2, of the 55 gm. of granulated lead, 15 were first placed in the scorifier and hollowed out to receive the sulphides, which should not touch the sides or bottom of the scorifier. The remaining 40 gm. were placed on the top of the sulphides.

In No. 4, the sulphides were first treated with nitric acid, and the silver was precipitated from the filtrate with hydrochloric acid and filtered off—this filter, with its contents, then being added to

the filter containing the residue undissolved by nitric acid. The two filters and their contents were then dried and assayed.

For assaying cupels, pulverize the cupel through a 30-mesh screen. Use 30 gm. fused borax, 30 gm. litharge and 30 gm. of sodium carbonate and cover with salt. Use a slow fire until nearly fused. If the crucible tends to boil over, throw in 20 to 25 gm. salt. For slags, all the slag from the assay is saved and pulverized through a 20-mesh screen, and mixed with flux used in experiment No. 4.

TABLE No. 713.

EXPERIMENTS IN ASSAYING "SULPHIDES."

Number of Experiment.	Method of Assay Used.	Weight of Sulphides A. T.	Weight of Fluxes Used.						Uncorrected Assay Value oz. per ton.	Silver in Slag per cent.	Silver in Cupel per cent.	Total in Slag and Cupel per cent.	Corrected Assay Value oz. per ton.
			Litharge, gm.	Sodium Carbonate, gm.	Borax, gm.	Potassium Cyanide, gm.	Charcoal, gm.	Granulated Lead, gm.					
1	Scorification	0.1			5.			55.	11595	0.28	1.04	1.32	11750
2	Scorification	0.05			5.			55.	11440	0.38	.48	0.86	11538
4	Crucible, with nail	0.20	22.	12.5			0.7		11482	0.14	.31	0.45	11533
9	" no nail ..	0.05	15.			25.			11399	0.13	.12	0.25	11427
3	" with nail	0.1	22.	12.5			0.7		11272	0.27	1.09	1.36	11425
5	" no nail ..	0.20	88.	11.					11375	0.10	.08	0.18	11395
8	" no nail ..	0.05	77.	5.5			0.6		11252	0.09	.12	0.21	11275
6	" no nail ..	0.20	25.			25.			11092	0.20	.19	0.39	11135

As shown in Table No. 713, the method by scorification, using one-tenth of an assay-ton, gives the highest "uncorrected" as well as the highest "corrected" assay, but with the exception of No. 3, carried the most silver (1.32 per cent.) into the slag and cupel. Nos. 5 and 8, which contain the most litharge, (77 and 85 gm.) have the least silver in the slag and cupel (0.2 per cent.); Nos. 6 and 9, with cyanide of potassium, have a little more silver (0.32 per cent.) in the slag and cupel. No. 3 is a common crucible-assay, made with a nail, but only 22 gm. of litharge. The slag and cupel contain the most silver (1.36 per cent.) of all the methods.

§ 714. *Assaying of the Sulphides for Gold.*

In assaying for gold, the sulphides should always be first treated with concentrated nitric acid, and only the residue assayed. Without this preliminary treatment, the amount of gold obtained will usually fall considerably short of the true amount. The following is the exact method:

Weigh out into a beaker (600 to 800 c. c. capacity) 1 to 3 assay-tons of sulphides. Cover with a watch-glass and add 200 c. c. of C. P. nitric acid, at the rate of 6 or 8 c. c. at a time. Set the beaker in a moderately warm place on the sand-bath. After action has nearly ceased, add a few more c. c. of acid. If action begins again, continue to add the acid until it ceases. Dilute with about 400 c. c. of pure hot water; let stand about an hour, and filter off the undissolved residue. Dry the residue and filter paper, and burn the latter. As the residue will contain considerable silver undissolved, no more need be added on account of the gold.

§ 715. *Testing Ores regarding their Fitness for Lixiviation.*

The metallurgist is often called upon to give an opinion regarding the treatment of ores by various processes. In order to determine whether an ore can be treated with profit by lixiviation or not, he has to investigate the following points:

1st. Whether the ore can be lixiviated raw, or requires roasting.

2d. How fine the ore has to be pulverized.

3d. The percentage of salt required for chloridizing-roasting.

4th. The loss of silver in roasting by volatilization.

5th. The difference between lixiviation with ordinary and extra-solution.

6th. The quantity of base metals dissolved by hyposulphite solutions.

In case the ore can be treated raw, the investigation is simple enough, the fineness to which the crushing must be carried being the principal point at issue. If sampling-works with rolls are available, the crushing should be done by rolls, and not by hand in a mortar.

Ores requiring roasting cause much more work. The following hints will be useful to beginners. Roasting-tests are best carried out in the muffle of an assayer's cupelling-furnace, in clay-dishes of about 4½ to 5 inches diameter, which will conveniently hold a charge of 3½ A. T. In order to arrive at results with the least amount of labor, it is best to commence with charges of ore that have been crushed so as to pass a No. 40 screen, and mixed with a maximum percentage of salt, which may be taken between 10 and 15 per cent., according to the character of the ore. Desulphurizing should be done slowly, and at a low heat, under constant stirring. As soon as glowing particles cease to appear, and the charge commences to swell, the temperature of the ore is raised to a dull red heat, and so maintained for half an hour. Another charge is roasted for one hour, to see if the chlorination of the silver is thereby increased. The roasted ore is weighed and assayed to determine the loss of silver in roasting. Samples are now lixiviated with ordinary solution, and by methods with extra-solution that are most adapted to the character of the ore. An examination of the raw ore will *a priori* indicate if the roasted ore may be expected to contain caustic lime, while leaching with water will definitely show whether the wash-water is alkaline, neutral or acid. After the proper method of applying the extra-solution has been determined, and also the influence of time in roasting ascertained, new roasting and lixiviation-tests are made with a reduced percentage in salt, until an economical limit is reached. These facts established, other roasting-tests are in order with ore crushed through No. 30 and No. 20 screens. If desirable, an investigation as to influence of temperature and time of roasting can be resumed. All the results are tabulated, so that they can be compared with each other at a glance.

It is not always possible to produce by muffle-roasting on a small scale the same effect that can be obtained by actual mill-work, especially if an ore is treated requiring banking-up on the cooling-floor for several hours in order to reach a higher chlorination of the silver. Hence, an unfavorable result is not always a proof that the ore is unsuitable for lixiviation. Should a failure of these experiments be due to the absence of a sufficient quantity of sulphurets to decompose the salt, new roasting-tests should be made with the admixture of from 1 to 3 per cent., or more, of

pyrites of iron. Of course, it should be ascertained beforehand if pyritic ores can be procured, in a given locality, at a reasonable cost. There are cases in which an ore gives highest results if the roasting is accomplished in the shortest time possible, with a charge spread out so that its thickness does not exceed one-eighth of an inch.

Finally, it is necessary to state that if the extra-solution gives an extraction only slightly in excess of ordinary lixiviation, this is not an absolute proof, in this respective case, of the inapplicability of the Russell process, since working on a large scale may change this relation.

In these roasting-tests the loss of silver by volatilization may be considerable, and thus depress the results of actual yield in silver. It is a well established fact that the loss of silver by volatilization in roasting in a Stetefeldt furnace is a minimum, and almost imperceptible, this loss being, principally, a function of time. I have demonstrated (see Trans. Am. Inst. of M. E., vol. xiv., page 336), that silver lost by volatilization in roasting is silver that would have been subsequently extracted if no such loss had taken place. In other words: Ore roasted in one case with considerable loss in silver, and in another without loss, would produce in both tailings of like value, provided the roasting had been carried to the same perfection in each furnace. Hence, a table of "Corrected Values" can be calculated from the results of muffle-roasting, to show what yield in silver may be expected in case the ore is roasted in a Stetefeldt furnace. This subject is of such importance, so little understood, and so much doubted by professional metallurgists, that a reproduction of the facts and observations, upon which my theory is based, will not be out of place. In 1880, I carried out, at the Ontario mill, a series of comparative experiments of roasting in the Howell and Stetefeldt furnaces. The mixture of ore and salt, from the same stamp-battery, was elevated, and equally divided by a mechanical arrangement, supplying both furnaces with identically the same material. The roasted ore from each furnace was amalgamated in pans, yielding bullion of the same fineness. A week's run gave the following results:

	Stetefeldt.		Howell.
Assay-value of raw ore		105.46 Oz. p. t.	
Percentage of salt		17.5%	
Value of roasted ore	82.42 Oz.		72.51 Oz.
Containing soluble salts	26.3%		25.8%
Value of roasted ore, minus soluble salts	111.91 Oz.		97.60 Oz.
Value of tailings	10.38 Oz.		10.14 Oz.

The loss of silver by volatilization in the Howell furnace was 12.7 per cent. greater than that in the Stetefeldt furnace, the loss in the latter not being perceptible. It will be seen that in this experiment the tailings, after amalgamation, were in both cases of almost exactly the same value. Basing calculations on the value of roasted ore minus soluble salts, we find that in the Howell furnace, 14.31 ounces of silver were volatilized. From the ore roasted in the Stetefeldt furnace, these 14.31 ounces (less 0.24 ounces) were extracted by amalgamation. Hence, I claim that my theory has received as good a practical confirmation as can be desired. Of course, what applies to amalgamation is also true for lixiviation.

By leaching an assay-ton of roasted ore first with water, and then with a solution of $1\frac{1}{2}$ or 2 per cent. concentration in sodium hyposulphite, lead can be precipitated from the solution by sodium carbonate, and silver and copper from the filtrate by boiling the acid solution. In the precipitate the copper is easily determined. In this manner an approximate idea can be obtained of what may be expected in working on a large scale. Also the wash-water may be tested for silver and copper. I annex a schedule for recording the results of roasting and lixiviation-tests.

SCHEDULE FOR RECORDING ROASTING AND LIXIVIATION-TESTS.

| Number of Charge. | | |

PART II.

THE LIXIVIATION-PROCESS

IN ITS

PRACTICAL EXECUTION.

CHAPTER XI.

The Lixiviation-Plant.

Improvements have been made within the last few years in the construction of lixiviation-plants; hence, this subject will be considered in all its details.

Dimensions.—The dimensions of apparatus for a lixiviation-plant are, of course, arbitrary. The sizes here given refer to a plant of large capacity.

Under-Structure of Vats and Tanks.—In setting up wooden vats and tanks,—which should be constructed as described in § 1102,—the bottoms must rest on joists as shown in the drawings, Figs. 12 and 13, leaving the staves entirely free. The joists are supported by a wooden frame with posts sufficiently high to give head-room underneath the vats and tanks. This arrangement is essential for observing and repairing leakage. The practice of setting vats and tanks upon a solid floor, or burying them in the ground, is severely condemned.

§ 1101. *Storage-Tanks for Stock-Solution, and Extra-Solution-Tank.*

Dimensions.—The dimensions of these tanks are 10 feet inside diameter and 9 feet inside height.

Of the three tanks shown in the mill-plan, one serves for preparing extra-solution, if the latter is not made on the charge in the lixiviation-vat.

Coil for Heating the Solutions.—Each tank is provided with a coil of $1\frac{3}{4}$ inch lead-pipe, 100 feet long, supported on wooden

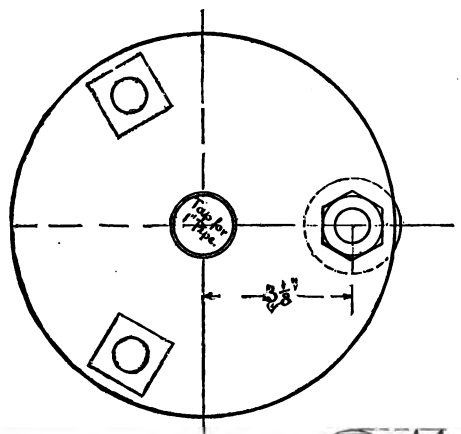


Fig. 20.

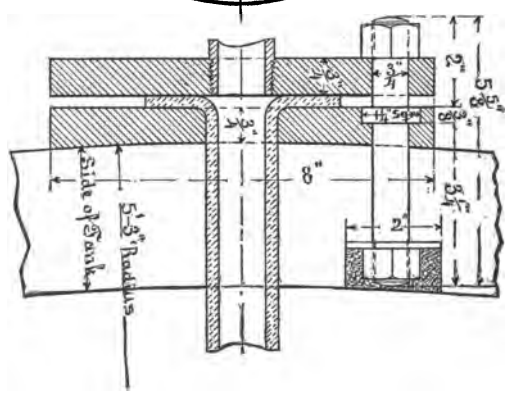


Fig. 21.

Connection of Lead- and Steam-Pipe.

blocks, about 4 or 5 inches above the bottom. The manner of connecting the lead-pipe with the iron steam-pipe, is given in Figs. 20 and 21. These coils serve for heating the solution by steam.

I would make the coil in the extra-solution tank of copper. Lead is not attacked by ordinary solution, but corroded by extra-solution. On the contrary, an ordinary solution dissolves copper, while an extra-solution, being already saturated with a cuprous salt, attacks this metal very little.

Tank-Connections.—These tanks have a triple connection with gate-valves, so that each can be used independently of the others; or two can be connected; or all three, which latter case, however, hardly occurs. This arrangement is necessary not only on account of the extra-solution, but because ordinary solution may be used, on the same ore, warm and cold, during different periods of the process, and because different ore-charges are not simultaneously in the same stage of manipulation.

Chemical-Boxes.—Above the tanks, and below the inlets of the stock-solution pumped up from the solution-sump, are wooden boxes, 2½ feet square and 1 foot high, with holes of 1 inch diameter in the bottom, serving to break the force of the stream of solution, which would otherwise disturb the small sediment of sulphides generally deposited at the bottom of the tank. The box above the extra-solution tank is used for receiving the chemicals—copper sulphate and sodium hyposulphite.

Outlets.—The solution-outlets of the tanks in the bottom are covered with small boxes, 6 inches high, having perforated tops. They not only prevent the solution from being drawn down below 6 inches from the bottom, and disturbing the sediment of sulphides that may be there, but also act as strainers for chips, etc., which would choke valves and pipes. The outlet of the extra-solution tank, however, is uncovered, so that a sediment of cuprous hyposulphite may be swept out.

Korting Ejector.—It is advisable to provide the extra-solution tank with a Korting ejector, so that the solution can be circulated if the chemicals in the box are not all dissolved after admitting the proper amount of ordinary solution.

Pipes for drawing off Sediment.—Besides the outlet for solution, each tank has another one, c'', which serves for cleaning out the sediment of sulphides.

Solution-Pipes.—There are two lines of pipes leading from the storage-tanks to the lixiviation-vats. One, p, is for cold ordinary solution, the other, q, for hot ordinary solution, and for

extra-solution. The time required for running out a charge of extra-solution is so short that an independent pipe-line for this purpose is not necessary. Dimensions of pipes are given in the plan. The iron pipes should be coated inside and outside with asphaltum varnish.

Valves.—All valves near the tanks are gate-valves; those on the pipe-lines above the lixiviation-vats are Jenkins' rubber-faced globe-valves.

§ 1102. *Lixiviation-Vats.*

Lixiviation-vats are generally constructed of wood, and then they are round. If built of brick, and lined with cement-concrete, they are rectangular.

Advantages of Large Size.—Lixiviation-vats of large capacity are more advantageous in every respect than small ones. So far as lixiviation, assay-office, and clerical work are concerned, a vat of 10 tons capacity requires as much attention as one holding a charge of 50 tons. Hence, in a mill of given capacity, the work can receive much more care if the charges are large. Although it may not be apparent at first sight, the amount of stock-solution, and consequently the quantity of sodium hyposulphite lost by atmospheric decomposition, is also reduced by conducting the process in vats of large capacity. Vats of 8 to 10 feet diameter and 3 to 4 feet depth, as they exist in old works, are entirely obsolete. For wooden vats constructive difficulties set a maximum limit as to diameter, which may be taken at 16 feet. The inside depth should not be more than 5 feet, if the tailings are shovelled out; and 7 feet, if they are removed by sluicing. Later on we will see what influence the depth of a charge has upon lixiviation, mechanically speaking.

Calculation of Capacity.—The capacity of a vat can be calculated if the weight of a cubic-foot of dry ore is known, and also its shrinkage after lixiviation has commenced, besides taking into account that the vat should only be filled within 12 inches of the top—the space needed for manipulating the solutions. One cubic-foot of crushed dry raw ore weighs from 75 to 110 pounds; one cubic-foot of dry roasted ore, from 55 to 90 pounds. The shrinkage for raw ores is from 10 to 18 per cent.; for roasted ores, from 12 to 24 per cent.

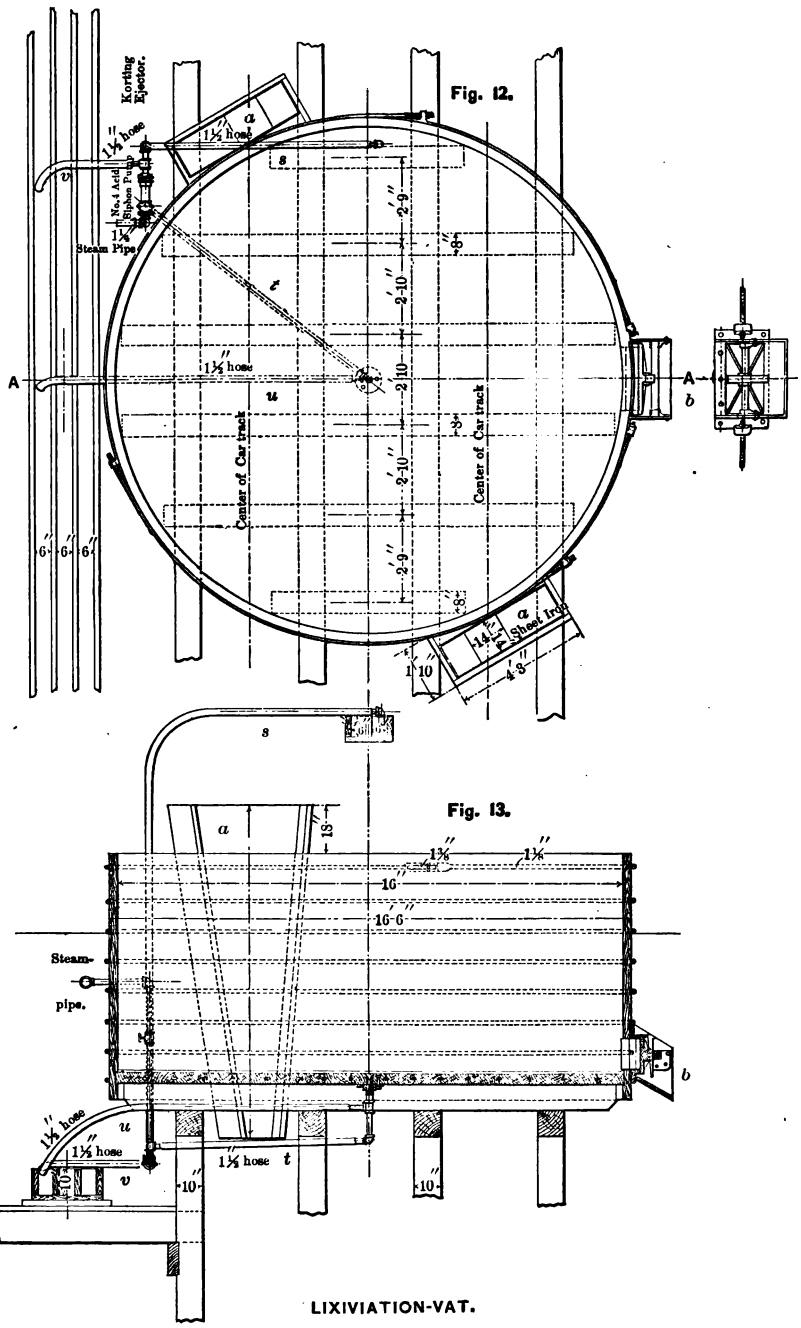
Lumber for Vats.—The best material for the construction of vats, and all tanks used in lixiviation, is California white cedar.

The Staves and Bottoms.—The staves, 3 inches thick, should be ordered cut to sweep of vat, and 9 inches longer than the inside height, but not gained for the bottom; and the latter should be without dowel-pins, cut to a diameter 2 inches greater than that of the finished vat. The gaining of the staves, 1 in. deep, is done by hand, leaving a chine of 6 in. below the bottom. The bottom pieces, 4 in. thick, are grooved and joined by a tongue $\frac{5}{8}$ in. by $1\frac{1}{2}$ in.* All joints of staves and bottoms must be fitted with precision, and are finally put together with a thick coat of white lead. Nails or screws must not be used. These precautions are imperative to obtain air-tight vats, so that the Korting ejectors may be used with effect.

Filters.—The false bottoms for the filter, and the latter itself, are prepared as follows: Wooden slats, $1\frac{1}{2}$ in. high and 1 in. wide, and separated 1 in. from each other, are fastened to the bottom of the vat by iron screws, bedded in thick white lead. The side of the slats next to the bottom is cut out in several places, $\frac{1}{2}$ in. deep and 3 in. wide, so that a passage for the solution is established everywhere. Between the ends of the slats and the staves of the vat a clear space, $1\frac{1}{2}$ in. wide, is left all round. A strip of wood, $1\frac{1}{2}$ in. high and 1 in. wide, previously cut by a saw in many places, and well soaked in water, so that it will bend easily, is now fastened round the slats, leaving an annular space, $\frac{1}{2}$ in. wide, between the strip and the staves. One thickness of stiff matting, covering the slats and the circular strip, but not the annular space, forms a foundation for the filter-cloth proper. The latter, No. 8 canvas-duck, is cut to a diameter 6 in. greater than the inside of the vat, so that the ends can be pressed into the annular space described above, and kept in position by forcing down a $\frac{1}{2}$-in. rope. Figs. 12, 13, 14 and 15, show all details. The filter does not require to be removed oftener than once in 6 or 8 months.

Hoops.—The hoops, made of $1\frac{1}{8}$ in. round iron, with threaded

* Mr. A. J. Corcoran, an experienced tank-builder of New York City, claims that a coat of white lead between the joints of staves and bottoms is more injurious than beneficial; also that dowel-pins are better than tongues in joining the bottom pieces. According to Mr. Russell's experience, however, the bottoms of such vats are not sufficiently tight if a vacuum is created below the filter by means of an ejector.



LIXIVIATION-VAT.

ends passing through forged lugs, are tightened by hexagonal nuts.

Troughs for Charging Solutions.—Above the lixiviation-vats runs a double wooden trough. One compartment has divisions between each two vats, and conducts water or lixiviation-solution to the vats through a box with perforated bottom underneath the trough. This box serves both for breaking the force of the water or solution-current, and for receiving the chemicals if extra-solution is made in the vat. The other compartment, with a plugged hole over each vat, is without divisions. Its principal object is to convey special extra-solution (see § 1405) from one vat to another. It can also be used for conveying extra-solution from the extra-solution tank. Alongside of the trough run the solution-pipes from the storage-tanks, described in the previous paragraph, and also a water-pipe.

Solution-Outlet and Korting Ejector.—Each vat has in the center of the bottom, below the filter, one outlet for solution. It consists of a threaded cast-iron flange, bolted to the bottom. The bolts pass through the bottom, and are, with their heads, imbedded in white lead. The details of these flanges, and the manner of connecting the two hose to them, are given in Figs. 12, 13 and 16.

The connections only are of iron gas-pipe, coated inside and outside with asphaltum varnish. The pipes are six-ply rubber steam-hose, the ends of which can be closed by wooden plugs. Steam-hose is used because more durable than ordinary hose; it neither kinks, sags, nor wears out at the ends by continual plugging. The hose, *u*, leads directly to the triple trough in front of the vats, while *t* connects the outlet with the Korting ejector. The latter discharges either into the triple trough by opening the hose, *v*, and closing the end of *u*, or returns the solution to the vat through the hose, *s*, after opening the valve in the perpendicular part of *s*, and closing the ends of *v* and *u*. It will be seen that this arrangement permits filtration at its natural rate through the hose, *u*, or increase of the rate of lixiviation (See § 1302), by means of the Korting ejector; or circulation of the solution. (See § 1201.) The hose, *t*, should be slanting downward, without sagging, and the Korting ejector must be so connected with it that the suction-opening is upward. Otherwise, the ejector works intermittently.

Triple Trough in Front of the Vats.—The horizontal triple trough in front of the vats serves the following purposes: One compartment, *k*, takes the concentrated silver-bearing first wash-water to the wash-water precipitating-tanks; another, *j*, conveys the lixiviation-solution to the lead or silver precipitating-tanks; the third, *l*, is used for running wash-water to waste. The troughs *k* and *j* have opposite each precipitating-tank plugged outlets formed by nipples of 2½-inch gas-pipe. These project about ¾ of an inch above the bottom of the trough, so that sediment formed there is not carried to the precipitating-tanks. Connection with the precipitating-tanks is made as shown in the drawing.

Tailings-Shoot.—If tailings are removed by shovelling they are discharged through the tailings-shoot, *a*, dropping into a car underneath the vats.

Sluice-gate.—A gate, *b*, for discharging the tailings by sluicing, is fastened to the side of the vat by iron bolts bedded in white lead. Its bottom should be ¼ of an inch below the surface of the filter. The gate proper is covered with a sheet of rubber, which protects the iron, and, at the same time, forms a gasket. The cast-iron frame is protected by asphaltum varnish. The manner of passing the lower hoop by the sluice-gate is shown in Fig. 12.

Mr. O. Hofmann, in his paper on trough-lixiviation, describes a very ingenious discharge for a lixiviation-vat, placed in the center of the bottom. At the same time, the filter is constructed in the shape of a funnel, with a slight inclination from the staves to the center. Such a discharge is, no doubt, a "*conditio sine qua non*" for his trough-lixiviation. For ordinary vat-lixiviation, a sluice-gate at the side of the vat is much simpler, and answers the purpose in every respect. Hofmann's central discharge makes the construction of the filter complicated.

Painting the Vats.—Finally, the vats are covered outside with three coats of white lead paint, the last one being of dark color, such as will not show dirt. This applies also to tanks and sumps. If the lumber is perfectly dry, several coats of paint inside of the vat will be beneficial. All troughs and boxes are painted with asphaltum varnish.

Vats of Cement-Concrete.—I am in favor of trying the experiment of constructing vats lined with cement-concrete. The walls of these vats could be erected of brick, well anchored with rails,

bolts, and buck-straps, and the lining of concrete built up after the walls are in place. To prevent dislocation or cracking of the walls, bottom, and lining, it is, of course, essential that the whole structure should rest upon a solid foundation of arches, the latter giving, at the same time, access to the discharge-pipes of the vats below the filters. These vats, being rectangular, would have more capacity in the same space than round ones, and could be built of any desired size.

§ 1103. *The Precipitating-Tanks.*

The tanks for precipitating lead as carbonate, or the precious metals, copper, and lead as sulphides from a lixiviation-solution, and silver as sulphide from the first wash-water, are all of the same construction. A tank specially adapted to precipitation of silver as cement-silver from the first wash-water will be described in § 1404.

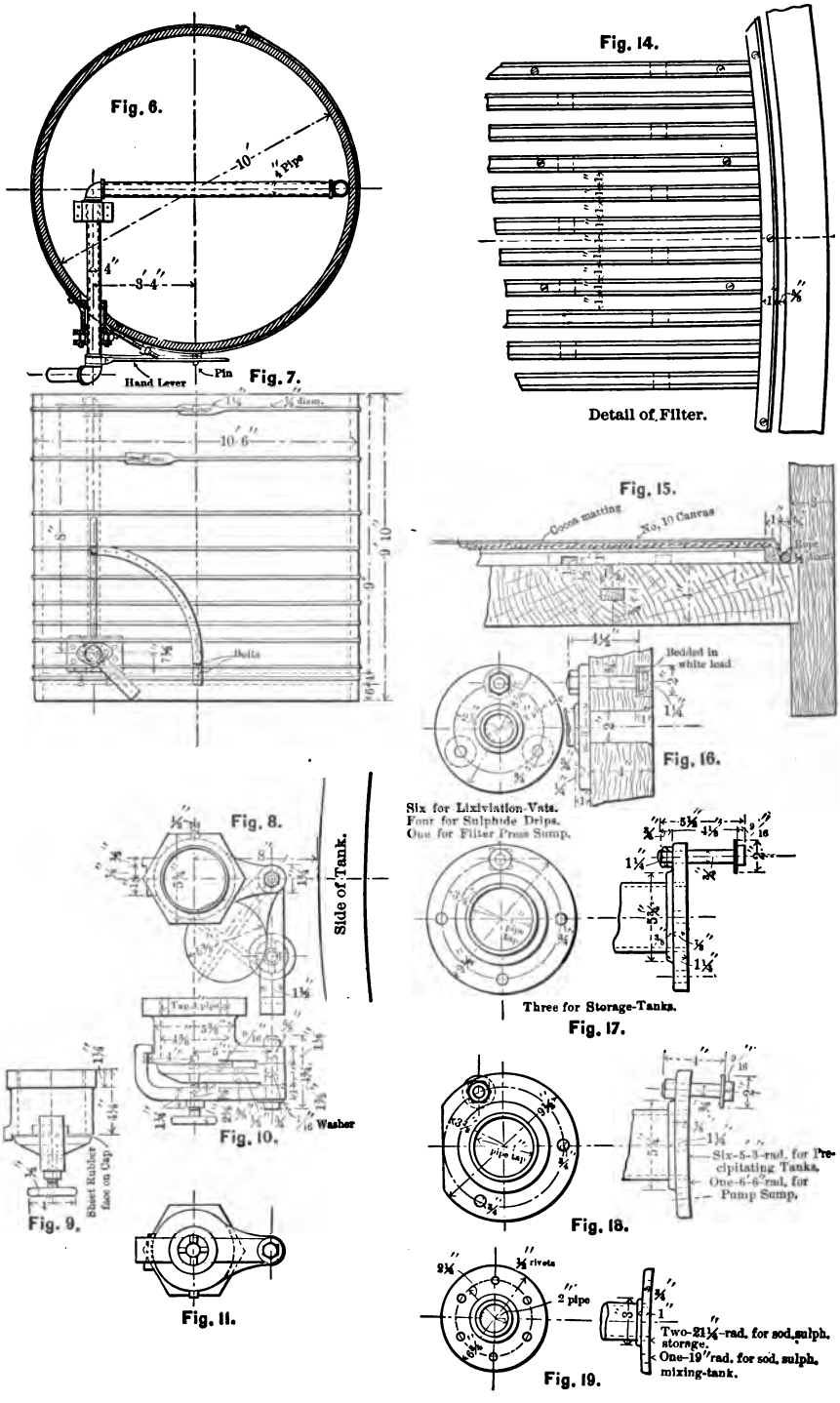
Dimensions.—The precipitating-tanks are of the same size as the tanks described in § 1101, namely, 10 feet diameter and 9 feet depth.

Outlets for Solution.—The construction of the outlets for solution is illustrated in Figs. 6 and 7. After settling the precipitate, the opening of the swinging pipe is gradually lowered, thus withdrawing the clear solution. The gas-pipe should be well coated, inside and outside, with asphaltum varnish.

Outlets for Precipitates.—For discharging the precipitates, the tanks are provided, in the rear, with an outlet in the side, close to the bottom. It is constructed like a molasses-gate, see Figs. 8, 9, 10 and 11. In the drawing of the tank itself, Figs. 6 and 7, this outlet is not shown. The gate is screwed to the gas-pipe nipple with cast-iron flange, Fig. 18. All parts are covered with asphaltum varnish.

Oars for Stirring.—For stirring the solution during precipitation, an oar of ash, 16 feet long, is provided. The platform for stirring is on a level 6 inches below the top of the tank.

Mechanical Stirrers.—Mr. O. Hofmann, who prefers precipitating-tanks of small dimensions, 8 to 9 feet diameter and 6 feet depth, has introduced mechanical stirrers at the North Mexican Mill. I advocated them myself many years ago. Mr. Russell claims that mechanical stirrers are entirely superfluous. Data are



DETAILS OF LIXIVIATION-PLANT.

Figs. 5, 6, and 7, Precipitating-tank; Figs. 8, 9, 10, and 11, Gate for discharging sulphides; Figs. 14, and 15, Detail of filter; Figs. 16, 17, 18, and 19, Discharge-pipes for the various tanks.

at present lacking to decide if there is really sufficient advantage in mechanical stirring to go to the expense of providing the necessary apparatus. It will be noted, in following this description of plant, that power transmitted by shafting and belts is not at all required in the lixiviation-building.

Trough for Solution and Wash-Water.—In front of the precipitating-tanks is a horizontal trough. It has a division between the third and fourth tank, so that clear hyposulphite solution can be run to the solution-sump, D, in one direction, and the clear wash-water, after precipitation of the silver, out of the mill, in another direction.

Trough for Sulphides.—The trough in the rear of the precipitating-tanks has 4 feet drop in 100 feet length. Through it the sulphides are transferred to the sulphide storage-tank, E.

Precipitating-Tanks for Lead.—If lead is precipitated as lead carbonate, separate tanks must be provided for that purpose, from which the clear solution is decanted, and transferred to the silver precipitating-tanks by Korting ejectors. In addition to this, there should be a separate storage-tank for lead carbonate. The same pressure-tank and filter-press, however, can be used.

§ 1104. *The Storage-Tank for Sulphides.*

It is 12 feet 6 inches in diameter and 6 feet deep. An outlet in the side, close to the bottom, or in the latter itself, connects it with the pressure-tank. The pipe discharging the sulphides into the pressure-tank ends near the bottom of the latter, and has a hose-coupling which is only connected when the pressure-tank is charged.

§ 1105. *The Solution-Sumps.*

The main solution-sump, to which the pump is attached, is 12 feet 6 inches in diameter and 6 feet deep. Its outlet in the side, close to the bottom, connecting with the pump, is covered by a straining-box, as described for the storage-tanks.

The small solution-sump, below the filter-press, receives the drainage from the latter, and leakage from the pump. It is provided with a Korting ejector which raises the solution to the upper

sump. Both sumps have in the bottom pipes for drawing off sediment.

Fig. 22.

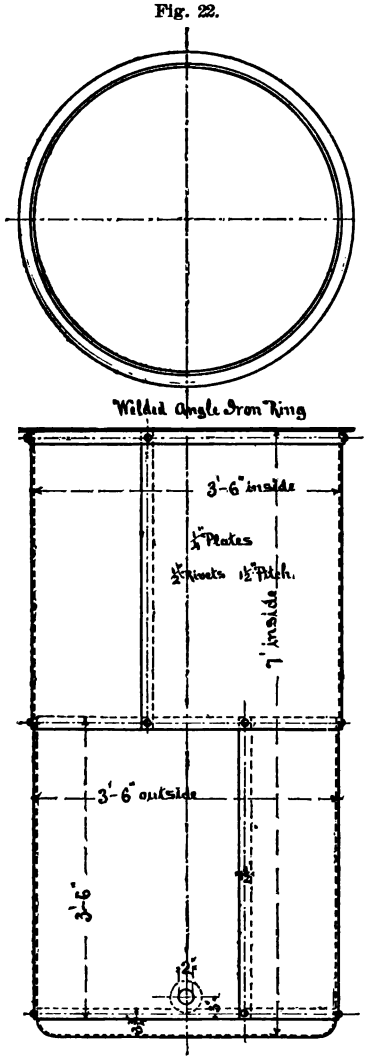


Fig. 23.
Sodium Sulphide
Mixing Tank.

Fig. 24.

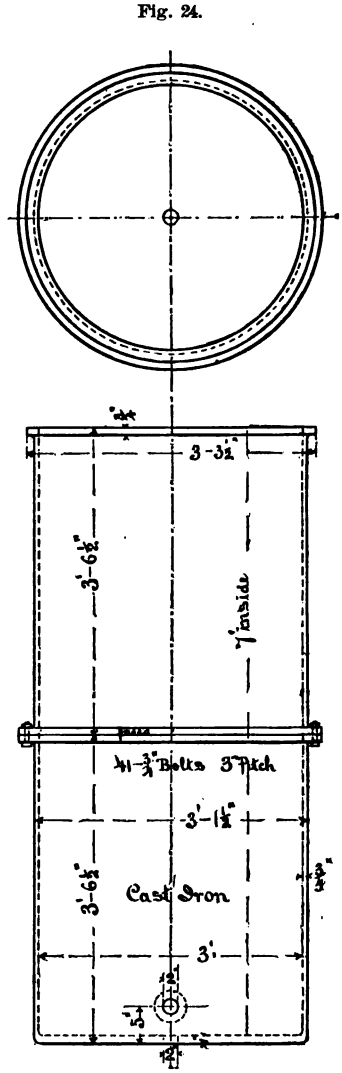


Fig. 25.
Sodium Sulphide
Storage Tank.

§ 1106. *The Sodium Sulphide Tanks.*

The tank, I, Figs. 22 and 23, for the preparation of sodium sulphide, is of cast-iron, in two sections. The sections are bolted to-

gether, and the joints leaded. Melted lead is poured on the bottom, 1 to 1¼ inches thick. A steam-pipe for heating the lye reaches within 1½ inch of this lead plate.

The two tanks, J, J, (see Plates I. and II.) figures 24 and 25, for the storage of sodium sulphide solution, are of ¼ inch boiler-iron. The flanges on top, shown in the drawing, are not necessary.

It will be noticed (see Plates I. and II.) that there is a partition forming a small room around the tank for preparation of sodium sulphide. This is necessary because in breaking up a drum of caustic soda, pieces are likely to fly, and get in the storage-tanks and vats.

Pipe-Lines.—These tanks are connected with a pipe for conveying stock-solution from one of the storage-tanks. A pipe-line leads from the sodium sulphide storage-tanks to all the precipitating-tanks. Where the pipes turn at right angles, they should have crosses, and not elbows. If the sodium sulphide solution is used very concentrated, the pipes may be stopped up by crystallization of the solution in cold weather.

§ 1107. *Tanks for Sodium Carbonate Solution.*

These tanks are of the same size and construction as the storage-tanks for sodium sulphide solution.

§ 1108. *The Korting Ejector.*

The modification of the Korting ejector used here is known as the "acid syphon pump." Its construction is illustrated by figure 26.

The nozzle of the steam-pipe is of platinum so that it will resist corrosion, keeping the opening of precisely the same diameter.

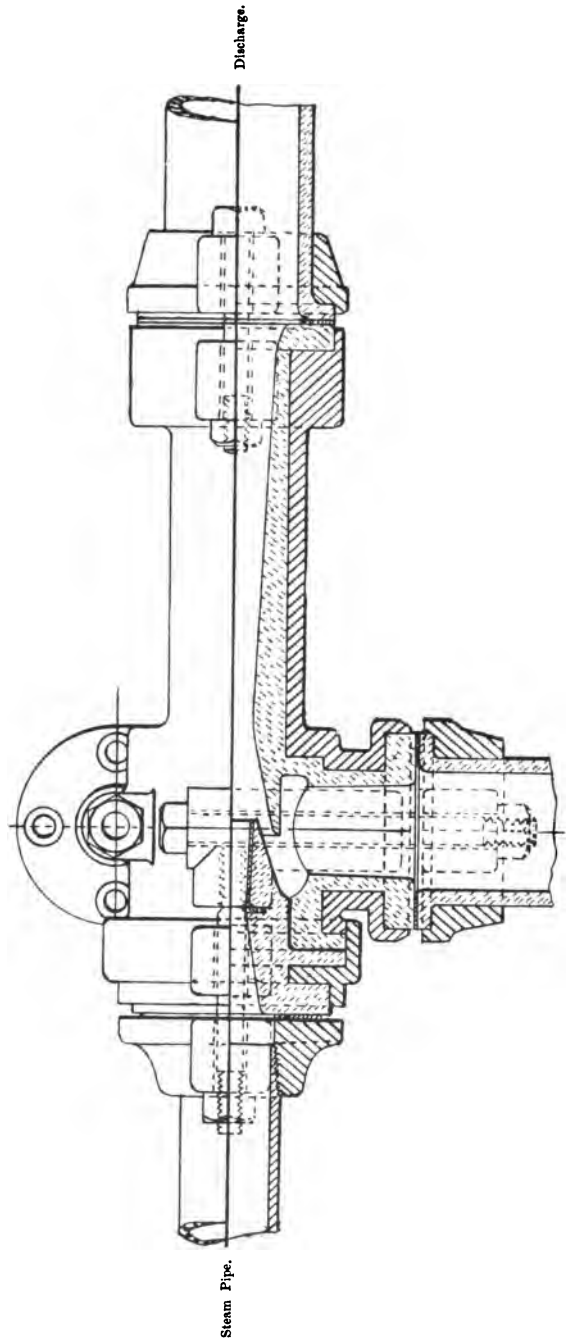


Fig. 26. The Korting Ejector.

The Korting ejector is made in various sizes. For lixiviation-vats of 12, 14, and 16 feet diameter, Nos. 2, 3, and 4 are used respectively. For effective work of the ejector, it is absolutely necessary that the steam should be of about 150 pounds pressure, and dry. Hence, the steam-pipes should be well wrapped with a non-conductor. If this precaution is neglected, a great deal of steam is condensed for little work done, and the lixiviation-solutions are perceptibly diluted. How the ejector is arranged in connection with a lixiviation-vat, has already been described in § 1102.

§ 1109. *Pumps for Raising the Stock-Solution to the Storage-Tanks, and for Sluicing the Tailings.*

The construction of pumps for raising the stock-solution from the solution-sump to the storage-tanks, presents some difficulties regarding the choice of material of which the parts of the pump, coming in contact with the liquor, are made. The solution always contains some silver and copper, and may be slightly acid, if a caustic solution has been neutralized with sulphuric acid, after precipitation. Under the circumstances, none of the base metals resist the corrosive action of the solution, lead not any better than iron, brass, or bronze. Although this action is quite slow, an ordinary pump commences to show signs of destruction after some time, much sooner in the valves than elsewhere. So far as chemical action is concerned, a lining of the cylinder, plunger, and valves with hard rubber would obviate this difficulty. Manufacturers of pumps, however, whom I have consulted about this subject, claim that construction with hard rubber is not only difficult, but that this material does not wear well. This brings up the question, if it would not be best to abolish pumps altogether, and use compressed air for raising the solution. This is done, for instance, in oil-refineries for elevating sulphuric acid. The pressure-tank receiving the solution could be made of iron, and coated with asphaltum varnish, which offers sufficient protection. The same air-compressor would also serve to work the pressure-tank for sulphides in place of steam. There would be no more difficulty in arranging a solution pressure-tank in connection with the

solution-sump than in connecting the sulphide storage-tank with the pressure-tank feeding the filter-press. (See § 1606).

The fire-pumps for luicing tailings should be a Knowles' double cap pattern, with 10-inch steam- and 5-inch water-cylinder, 12 inches stroke, and 4-inch suction and discharge.

If a solution-pump is used, it should be a No. 6½ Knowles' double plunger, with 8-inch steam cylinder, 5-inch plunger, and 12 inches stroke, and a 4-inch suction and discharge.

§ 1110. *The Filter-Press, and the Pressure-Tank.*

The Filter-Press.—Since the introduction of the filter-press, all other modes for separating and collecting precipitates from solutions have become obsolete. The filter-press consists of a series of

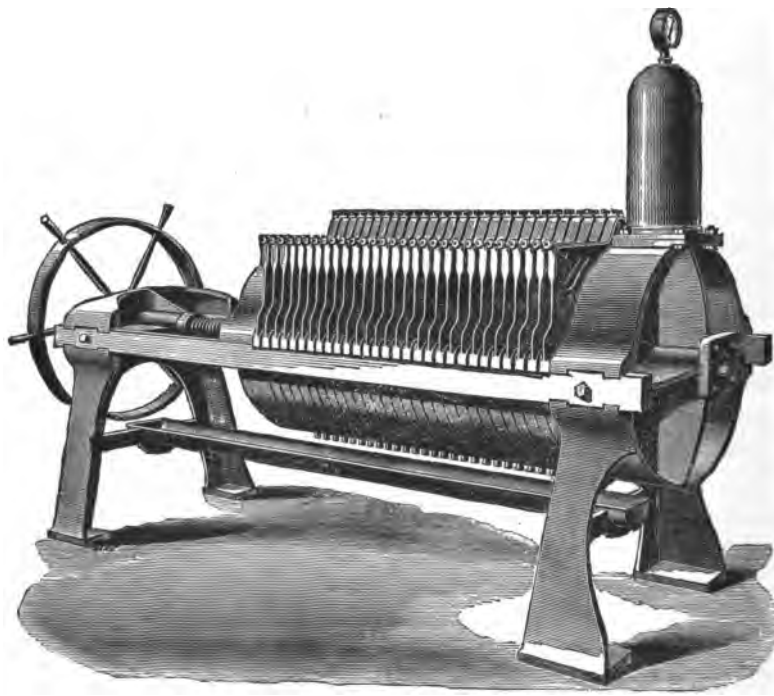


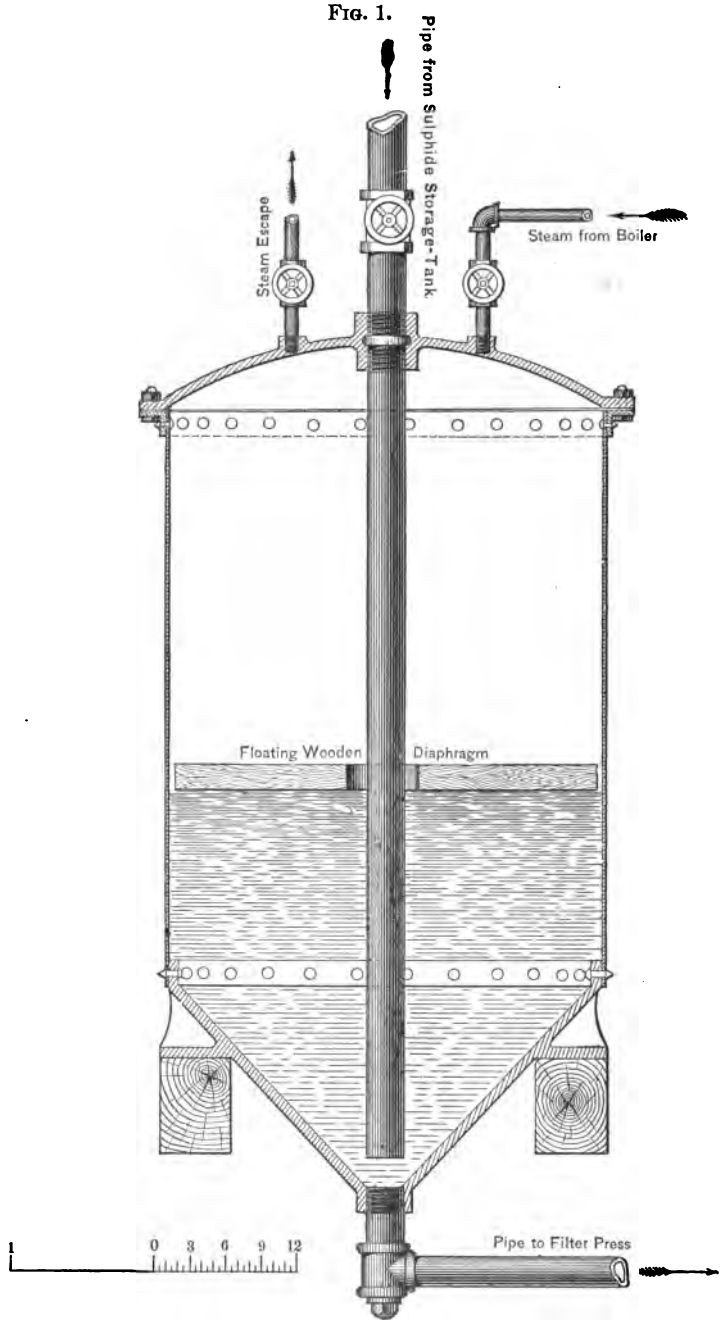
Fig. 27.—Johnson's Filter-Press.

round or square plates, made of cast-iron, lead, hard wood, or other suitable material, having projecting lugs so that they can be supported in a press-frame, in juxtaposition, face to face, and screwed up tightly between the head and follower of the press. The plates are concave on each side, while the projecting, truly-faced rims maintain the plate surfaces at distances corresponding

to the depth of two rims. Faced rings can be inserted between the rims of two plates to increase the distance between their surfaces. The plates, provided with channels communicating with outlets at the bottom, are covered with suitable filtering-cloth. Thus, the spaces between the cloth-lined plates form chambers into which the semi-fluid material to be filtered is forced under pressure. A passage, also lined with cloth, is formed through each plate, so that there is communication between all the filtering-chambers. A pair of cloths are used to cover each side of a plate, sewn together round a center-hole, corresponding to the opening in the plate. It is obvious that on folding one cloth, passing it through the hole in the plate, and then opening it out, both surfaces of the latter will be covered. The hooks on the plates, to which the filtering-cloth is fastened, are movable by means of screw spindles, so that slackness of the filters can be taken up in a radial direction, thus insuring tight joints between the plates. The material for the filters, which must combine strength, durability, and closeness of texture, is specially manufactured and called "*lambskin.*" When the solution to be filtered is forced into the chambers of the press the liquid is driven through the cloth, and flows away through the outlets at the bottom of the plates, while the solid matter is arrested in the chambers. Finally, when the solid matter fills every chamber completely, the operation of charging is suspended. This is indicated by the opening of a safety-valve at the maximum pressure, say 150 to 180 pounds per square inch, for which the press is designed. Now, the press is unscrewed, the plates are separated, and without removing the filters, the chambers are emptied, their contents being in the form of solid cakes with more or less moisture, according to the character of the precipitate, and the pressure used in charging. The latter is indicated by a pressure-gauge. It does not take more than one minute to unload each chamber.

For lixiviation-mills, I recommend Johnson's filter-press with 24 round plates of 15 to 18 inches diameter. The cakes formed in the chambers have a thickness of 1¼ inch. Some distance-rings may be ordered to increase the thickness of the cakes, and the capacity of the press. The plates are of cast-iron, painted with asphaltum varnish. The latter should be occasionally renewed, so that the lixiviation-solution does not corrode the iron.

FIG. 1.



PRESSURE-TANK.

A filter-press may be charged either by a plunger force-pump or by a pressure-tank. For lixiviation-mills, the latter system deserves preference, as found by practical experience. It is difficult to prevent foreign matter, like chips of wood, etc., from entering the sulphides, and this interferes with the proper working of the pump-valves. Besides, what has been said in § 1109 about pumps, applies also to this case.

The Pressure-Tank.—It is illustrated in Fig. 1, and does not require further description. In § 1604 a detailed account of its manipulation will be found. According to the drawing, Fig. 1, taken from Mr. Daggett's paper, the lower, funnel-shaped part of the tank is made of cast-iron. This is bad construction; steel-sheet or boiler-iron should be used. Riveting steel or wrought-iron to a casting, there is danger of cracking the latter.

§ 1111. *The Drying-Chamber for Sulphides.*

Figs. 2, 3, 4 and 5 illustrate a drying-chamber for sulphides, heated by steam, as constructed at the Cusihuiriachic mill, Mexico. The drawing is taken from Mr. Daggett's paper.

§ 1112. *Steam-Boiler.*

It is advisable to provide in a lixiviation-mill a separate steam-boiler for the following purposes:

1st. To heat the lixiviation-solution in the storage-tanks.

2d. To heat the concentrated first wash-water from which silver is to be precipitated by scrap-iron.

3d. To run the fire-pump and the solution-pump, or the air-compressor.

4th. To heat the sulphide drying-chamber.

5th. To heat the caustic soda or the caustic lime solution in preparing sodium or calcium sulphide.

6th. To run the Korting ejectors.

7th. To supply the necessary pressure for working the tank designed to charge the filter-press.

For the first two purposes it is immaterial whether the steam is of low or high pressure, but, for the last four, dry steam of high pressure is most desirable. Hence, a boiler of the tubular pattern

should be selected which will safely sustain a pressure of 150 to 180 pounds to the square inch. In using a high pressure, much less steam is consumed by the Korting ejector, and, consequently, the

FIG. 2.

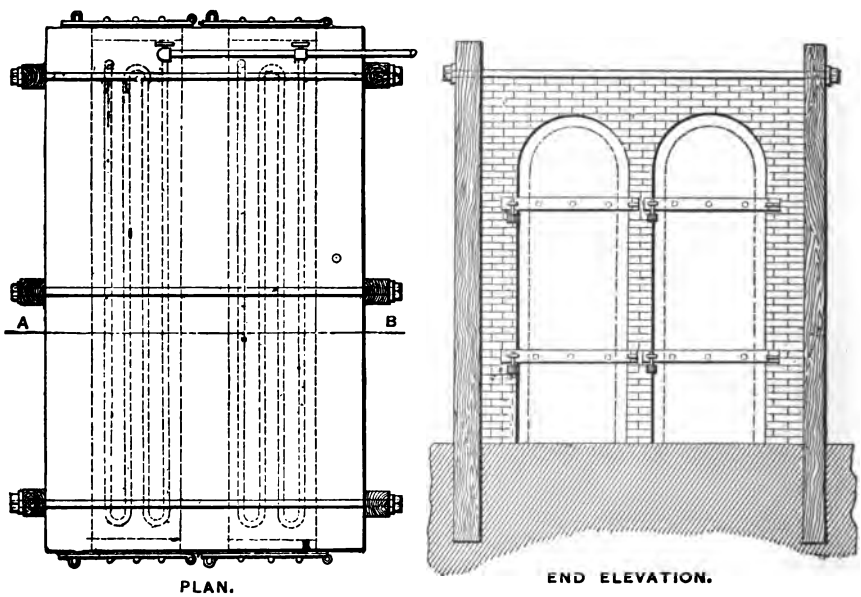
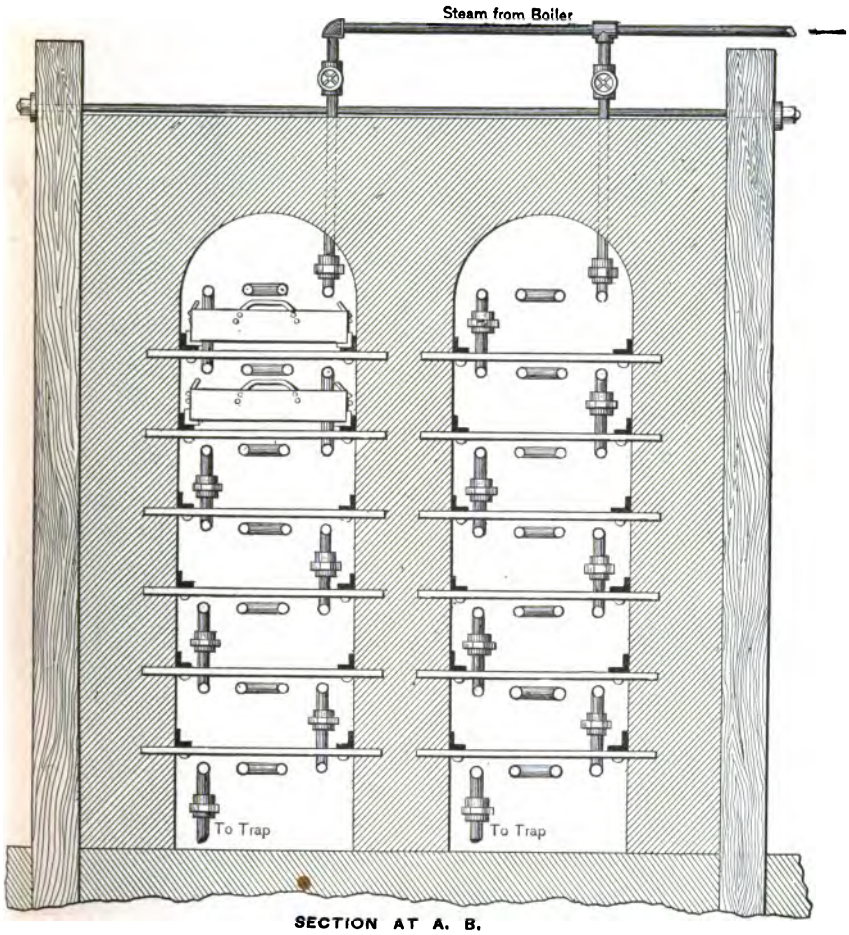


FIG. 3.

lixiviation-solution is less diluted; sodium and calcium sulphide are more quickly prepared; the cakes of the sulphides in the filter-press are more solid, and contain less moisture; the drying of the

FIG. 4. DRYING-CHAMBER FOR SULPHIDES.



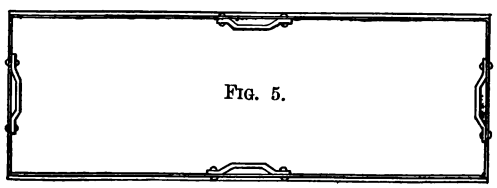
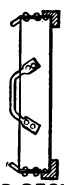


FIG. 5.

PAN FOR SULPHIDES.



CROSS SECTION

sulphides is accomplished in less time. An estimate of the necessary boiler-capacity can be obtained from the fact that in order to elevate the temperature of 1000 cubic feet water, in one hour, 1° F., a boiler-capacity of about 2.15 H. P. is needed.

I will illustrate this by an example. The mill is to treat 60 tons ore per day, one ton of ore requiring 100 cubic feet of hyposulphite solution of 120° F. It is assumed that the temperature of the solution, after returning it to the storage-tanks, does not fall below 80° F. It is further assumed that one ton of ore will yield 15 cubic feet wash-water from which silver is to be precipitated by scrap-iron, and its temperature is to be raised from 60° to 200° F. A calculation will show that the temperature of 250 cubic feet solution has to be elevated, in one hour, 40° F., requiring a boiler-capacity of 21.5 H. P. The temperature of 33.3 cubic feet wash-water has to be raised 140° F. in one hour, equivalent to a boiler-capacity of 10 H. P.

The quantity of steam consumed by the Korting ejectors, pumps, the pressure-tank, the sulphide drying-chamber, and in the manufacture of sodium sulphide, is comparatively slight, so that a total boiler-capacity of 40 H. P. would be sufficient for all purposes.

The plant for refining the sulphides will be described in § 1606. For refining the sulphides, a separate building should be erected.

§ 1113. *The Arrangement of the Lixiviation-Plant.*

The arrangement of the lixiviation-plant is shown in the drawings, Plates I. and II., and does not require description.

Index to Parts of Plant in Drawings, Plates I. and II.

A. Lixiviation-vats.
B. Silver precipitating-tanks.
C. Wash-water precipitating-tanks.
D. Main solution-sump.
E. Storage-tank for sulphides.
F. Filter-press sump.
G. Storage-tanks for ordinary solution.
H. Storage-tank for hot ordinary solution, or for preparation of extra-solution.
I. Cast-iron tank for preparation of sodium sulphide.
J. Sheet-iron tanks for storing sodium sulphide solution.
K. Pressure-tank.
L. Filter-press.
M. Steam-dryer for sulphides.
N. Boiler.
O. Fire-pump for sluicing tailings.
P. Solution-pump.
Q. Main ore-track to lixiviation-vats.
R. Side-tracks to lixiviation-vats.
S. Side-tracks in chemical room.
T. Grade-line between boiler and steam-dryer.
U. Grade-line through sulphide storage-tank.

a. Tailings-shoot to cars.
b. Sluice-gate for tailings.
c. Tailings-sluice.
d. Chemical box.
e. Ore-car.
f. Ore-car truck.
g. Chemical car.
h. Special extra-solution trough over lixiviation-vats.
i. Solution-trough over lixiviation-vats.
j. Solution-trough to silver precipitating-tanks.
k. Wash-water trough to wash-water precipitating-tanks.
l. Waste-water trough.
m. Trough to conduct sulphides to sulphide storage-tank.
n. Trough to conduct solution to solution-sump.
o. Waste wash-water trough.
p. Solution-pipe to lixiviation-vats.
q. Hot ordinary solution or extra-solution pipe to lixiviation-vats.
r. Water-pipe to lixiviation-vats.
s. Hose for circulating extra-solution.
t. Hose from lixiviation-vat to Korting ejector.
u. Hose from lixiviation-vat to triple trough.
v. Hose from Korting ejector to triple trough.
w. Korting ejector.
x. Tailings-car.
y. Hose-connections for sluicing.
z. Waste-water pipe from triple trough.

a'. Pipes from triple trough to silver precipitating-tanks.
b'. Pipes from triple trough to wash-water precipitating-tanks.
c'. Gate for sulphides.
d'. Surface-discharge for precipitating-tanks.
e'. Sodium sulphide pipe.
f'. Pipe for precipitated wash-water.
g'. Connection between solution-sump and pump.
h'. Strainer around outlet.
i'. Drip-pipe from solution-pump.
j'. Pipe from pump to storage-tanks.
k'. Connection between sulphide storage-tank and pressure-tank.
l'. Connection between pressure-tank and filter-press.
m'. Korting ejector in filter-press sump.
n'. Pipe for returning ordinary solution to solution-sump.
o'. Small windlass for steam-dryer.
p'. Boxes over storage-tanks to prevent agitation of solution.
q'. Boxes and strainers around outlets.
r'. Lead-coils in storage-tanks.
s'. Drip from lead-coil.
t'. Triple connection between the storage-tanks.
u'. Main steam-pipe.
v'. Water-pipe from fire-pump for sluicing.
w'. Storage-room for caustic soda and sulphur.
x'. Storage-room for copper sulphate and sodium hyposulphite.
y'. Trap-door to filter-press sump.
z'. Gauge-valve on side of pressure-tank.

a''. Solution-pipe to sodium sulphide storage-tank.
b''. Pipe for draining off sediment from solution-sump.
c''. Pipe for drawing off sediment from solution storage-tanks.
d''. Korting ejector for circulating solution through chemical box in extra-solution tank.

CHAPTER XII.

The Solutions.

§ 1201. *Definition of Terms.*

An "*acid ore*" is a roasted ore producing a first wash-water which gives an acid reaction.

An "*alkaline ore*" is a roasted ore, containing caustic lime, which produces a first wash-water giving an alkaline reaction.

An "*arsenical alkaline ore*" differs from the last named one in that it contains an unusual amount of arsenates. The only example of this kind, so far observed, is the Yedras ore.

"*Circulating*" a solution means withdrawing it by means of a Korting ejector from below the filter of a lixiviation-vat, and returning the solution to the latter, so that it filters continuously and repeatedly through the ore.

"*The volume of solution to saturate a charge*" is the quantity of liquid that fills the space between the particles of ore.

It is expressed in cubic feet per ton of ore. If a charge of ore is already saturated with one liquid, and the latter is displaced by another, the volume required to saturate the charge is reduced from 6 to 10 per cent., because some of the former liquid remains.

§ 1202. *The Ordinary or Stock-Solution.*

Preparation.—The ordinary or stock-solution is made by dissolving a proper quantity of sodium hyposulphite in a measured quantity of water in the storage-tanks. As the weight of a cubic foot of water is 62½ pounds, this number of pounds of sodium hyposulphite, added to each 100 cubic feet of water, will produce a solution of one per cent. concentration, and so on.

Concentration in Sodium Hyposulphite.—Practical experience has demonstrated that it is, in most cases, not economical to work with a solution of higher concentration than 2 per cent., even if high-grade ores are treated. In that case, increased volume of the solution is made to do the work. With low-grade ores, a solution of 1 per cent., and one of less concentration, may be most profit-

able. The concentration of stock-solution, actually used for treating roasted ores from various mines, is as follows, viz: Yedras*, 1.8; Ontario, 1.4; Sombrerete, 1.0; San Antonio and Sierra Grande, 0.8; San Bartolo, 0.7 per cent. $Na_2S_2O_3+5aq$.

In starting up new works, it is best to begin with a $1\frac{1}{2}$ per cent. solution, and then diminish or increase its concentration according to circumstances.

Volumes of Stock-Solution used per Ton of Ore.—The following statement gives the volumes of ordinary or stock-solution used per ton of ore.

	Volumes of Stock-Solution, Cubic Feet.
Silver Reef tailings,	100
Bremen† "	9
Veta Grande "	70
Silver Reef raw ore	137
Sierra Grande roasted ore,	80
Yedras " "	100
Ontario, 1887–8 " "	196
Daly " "	180
Veta Grande " "	110
Ontario, 83–84, " "	190
Sombrerete " "	90
San Antonio "	100
San Bartolo " "	80
San Miguel " "	70

Change in Concentration by Continued Use.—The strength of the solution, after continued use, changes, and in most cases it becomes weaker. The weakening is caused:

1st. By admixture of wash-water which may precede, and always follows the solution.

2d. By condensation of steam from the Korting ejectors.

* According to latest information, the concentration of the stock-solution in sodium hyposulphite is 2.3 per cent.

† The small volume of solution used on the Bremen tailings is due largely to its strength (3 per cent) in sodium hyposulphite. Owing to the extreme slowness of the rate of lixiviation, a strong solution was used to shorten the time as much as possible. In nearly all the cases, except that just mentioned, less solution could probably have been used if it had been desirable to do so, but such a course would have made necessary a preliminary assay of the tailings before removal, which would have caused delay. In nearly all cases the tailings are removed before the assay of the tailings is known, a course which is nearly always allowable if lixiviation is systematically conducted.

3d. By decomposition of sodium hyposulphite through atmospheric influences. This is especially the case if extra-solution is used, or if roasted ores, containing cuprous chloride, are treated.

4th. By decomposition of sodium hyposulphite with copper sulphate in making extra-solution: For each pound of copper sulphate thus consumed, one pound of sodium hyposulphite is destroyed. It is absolutely false to assume that sodium hyposulphite, thus converted into tetrathionate, is again regenerated in the precipitating-tanks.

5th. By decomposition of sodium hyposulphite, if concentrated sodium sulphide solutions, with high precipitating-coefficients, are used.

Loss in Sodium Hyposulphite.—The table below gives the actual loss in sodium hyposulphite, treating roasted ores by the Russell process.

Acid Ores.		Alkaline Ores.	
Ontario	5 lbs.	Veta Grande	4.0 lbs.
San Antonio	7 "	Sierra Grande	2.6 "
Sombrerete	5.3 "	San Miguel	3.6 "
San Bartolo	3.0 "	Daly	1.5 "
Average	5.06 lbs.	Average	2.9 lbs.

It will be noted that a considerable difference exists between "acid" and "alkaline" ores. The loss with alkaline ores is less:

1st. Because, as will be seen in the next paragraph, less copper sulphate is used for making the extra-solution; hence, less sodium hyposulphite is converted into tetrathionate.

2d. The extra-solution for treating alkaline ores, as will be seen in the next paragraph, contains less copper sulphate than one of standard composition, and is in consequence more permanent.

3d. Cuprous chloride generally exists in acid roasted ores, which is dissolved by the ordinary solution first applied. In this way a weak extra-solution is formed, which deteriorates quicker than an ordinary solution without copper.

Gain in Sodium Hyposulphite.—On the other hand, the solution gains in strength by evaporation, but principally in precipita-

ting the sulphides of silver, copper, and lead (provided lead has not been previously eliminated by sodium carbonate), with calcium or sodium sulphide of low or normal precipitating-coefficients. The more silver, lead, and especially copper precipitated, the larger will be the quantity of hyposulphite salts added to the solution, so that cases may exist where all the loss in hyposulphite salts is not only compensated, but the solution may even increase in strength. This occurs frequently if base ores, after chloridizing-roasting, are treated by ordinary lixiviation. Where extra-solution is used, the stock-solution can never gain strength in hyposulphite salts, unless the quantity of copper sulphate consumed should be very small, and the silver should be precipitated by old and highly oxidized sodium or calcium sulphide.

Maintenance in Strength.—After the most advantageous concentration of the solution has been determined by experience, it should be maintained by constant addition of sodium hyposulphite. This is best done—if the Russell process is used—in making the extra-solution, as will be shown in the next paragraph.

Determination of Concentration.—The concentration of a stock-solution in sodium hyposulphite, after it has been in use, can not be estimated, even approximately, by its specific gravity, for reasons that are apparent. Hence, the chemical test, described in § 711, is the only one upon which reliance can be placed.

Concentration in other Salts.—The original stock-solution is subjected to other changes, outside of those mentioned above, in consequence of a gradual accumulation of sodium chloride, sulphate, and tetrathionate. It can easily be seen whence these salts are derived. Sodium chloride and tetrathionate do absolutely no harm. Sodium sulphate diminishes the solvent energy of the solution only slightly. Calcium-salts are introduced if gypsum is present in raw ore, or if caustic lime or calcium sulphate exist in roasted ore, or if calcium sulphide is used as precipitant for silver. In case the lead is precipitated by soda, the calcium is also precipitated, and no calcium-salts can exist in the solution.

The concentration of the solution in these salts, however, does not go on indefinitely, but reaches a maximum and then remains stationary. It can easily be seen that this effect is produced by the wash-water, which either precedes and follows, or merely fol-

lows the lixiviation-solution. In this operation both a loss and a dilution of the lixiviation-solution takes place.

Caustic Stock-Solution.—There is, however, another change in the condition of the stock-solution that is of great importance, namely, its assuming a caustic reaction. How seriously this condition of the solution affects the final result of lixiviation has been shown in § 218. Only silver-ores that carry a comparatively large percentage of antimonial and arsenical compounds may be benefited by treatment with a caustic solution. Whenever the stock-solution shows the presence of caustics, it must be neutralized with diluted sulphuric acid. The consumption of sulphuric acid per ton of ore may be from $\frac{1}{4}$ to $\frac{3}{4}$ pounds. It seems to me most convenient to add the sulphuric acid to a caustic stock-solution in the silver precipitating-tanks, immediately after the precipitation of the sulphides has been finished, and before the solution is returned to the storage-tanks. The reaction of the solution on litmus paper should be closely observed so that not more acid is consumed than is absolutely necessary. Copper sulphate also neutralizes the caustic.

Volume of the Stock-Solution in the Mill.—It is evident that the amount of stock-solution kept on hand and in circulation, should be reduced to a minimum. The quantity of stock-solution required for works of given daily capacity depends upon the time of lixiviating with extra- and ordinary solution, and upon the volume of solution necessary to saturate a ton of ore. This will be made clear by an example. Suppose the daily capacity is 80 tons; one ton of ore needs 9 cub. ft. of solution for saturation; the time of lixiviating with extra-solution is six hours, and with ordinary solution twenty-four hours. In that case, 100 tons of ore are constantly under treatment, and they require 900 cub. ft. of solution for saturation; besides, a certain quantity of solution stands above the charge in the vats, say 150 cub. ft.; one precipitating-tank is full and settling, and a second one is half-filled, say 750 cub. ft. for both; then there is a certain accumulation of solution in the sumps for the pump and filter-press, say 250 cub. ft.; finally, we must leave sufficient solution in the storage-tanks to make a charge of extra-solution of about 300 cub. ft., and in addition to that we should have not less than 300 cub. ft. more solution to supply the lixiviation-vats. This would give a total of at least 2400 cub. ft.

That works of large capacity require comparatively less stock-solution than those of smaller capacity can easily be seen.

In most cases the best results are obtained in lixiviating with an ordinary solution heated from 35° to 50° C., or 95° to 122° F.

§ 1203. *The Extra-Solution.*

Extra-Solution not Kept in Stock.—The extra-solution is not kept in stock, but made in charges as needed, immediately before use, for reasons fully stated in § 317. After passing the precipitating-tanks it is converted into ordinary solution.

Composition of Extra-Solution. — Extra-solution of standard composition is used for raw, acid, and alkaline arsenical ores, while alkaline ores are treated by extra-solution with less copper than is required for standard composition. In the latter case, the proportion of $CuSO_4+5aq.$, to $Na_2S_2O_3+5aq.$, varies between 1 to 3, or 1 to 4, and even extra-solutions much weaker in copper are profitably used, as will be seen later.

Rules for Preparing Standard Extra-Solution.—The stock-solution, used in the preparation of standard extra-solution, is generally of such low concentration in sodium hyposulphite, that it does not contain a sufficient quantity of this salt to form a solution of desired concentration in copper. Hence, in dissolving the copper sulphate a proper amount of sodium hyposulphite must be added. I will illustrate this by an example. We will suppose that ore is treated which requires 8 cub. ft. of standard extra-solution per ton; that the normal concentration of the stock-solution is 1½ per cent. in sodium hyposulphite; that 5 pounds of copper sulphate are needed per ton of ore. The 8 cub. ft. of stock-solution would represent only 7½ pounds of sodium hyposulphite; hence, not less than 3¾ pounds of this salt should be added for each 5 pounds of copper sulphate in 8 cub. ft. of solution, in order to produce a standard extra-solution. Of the 11¼ pounds of sodium hyposulphite now contained in 8 cub. ft. of solution, 5 pounds will be destroyed by the copper sulphate, leaving 6¼ pounds intact. After precipitating the extra-solution, an ordinary solution of 1¼ per cent. concentration would result, not considering either loss or gain in sodium hyposulphite by precipitation, or loss by oxidation of the extra-solution while in use. Before making standard extra-solution it should be ascertained whether the ordi-

nary solution is alkaline or acid. If alkaline, an extra solution will result which deteriorates rapidly, as shown in § 311. In the presence of too much acid, the temperature at which the solution decomposes upon heating is materially reduced. Commercial copper sulphate generally contains sufficient free acid to produce a permanent standard extra-solution from a neutral stock-solution. Extra-solutions with twice the amount of sodium hyposulphite required for standard composition are quite permanent, if neutral. Addition of a slight quantity of sulphuric acid or sodium carbonate to the stock-solution easily corrects these defects.

Modes of Preparation: First Method.—Extra-solution is either prepared in a tank or on the charge itself in the lixiviation-vat, in the empty space above the ore. The first method is principally used if raw ores are treated first with extra-solution, no other liquid having preceded. The necessary amounts of copper sulphate, together with sodium hyposulphite, put into a box with holes, placed above the extra-solution tank, are dissolved by ordinary solution turned on from the storage-tank. Before this is done, however, the tank below is filled with about two-thirds of the required amount of ordinary solution. In this way the strong copper solution meets at once a sufficient amount of sodium hyposulphite to prevent the precipitation of a canary yellow salt of cuprous hyposulphite. After the chemicals are dissolved, the contents of the tank are stirred to dissolve cuprous hyposulphite that may have been precipitated. If the extra-solution is to be used warm, it is best to heat the stock-solution to the required temperature before turning it into the extra-tank.

Testing Standard Extra-Solution.—A sample of standard extra-solution should be clear, and not become turbid if shaken with air in a partly filled flask. Turbidity would indicate the entire absence of free acid. I have shown, in Chapter III., III., that acid reaction with litmus-paper is by no means a proof that the extra-solution contains free acid. Hence, the litmus-paper test can not be relied upon. If the stock-solution, used in preparing the extra-solution, was not clear originally, the test described above should be made with a few c. c. of extra-solution after filtering. If the standard extra-solution turns dark when heated to 65° C., or 149° F., this would indicate the presence of too much free acid. The remedy in both cases has already been mentioned.

The Second Method.—The second method of preparing extra-solution may be used if the latter is preceded by wash-water or ordinary solution. The discharge-pipe of the lixiviation-vat is closed as soon as the liquid has sunk to the surface of the ore. Proper amounts of copper sulphate and sodium hyposulphite, placed in the perforated box above the vat, are dissolved by cold or warm stock-solution, which is added until the space above the ore, say 12 inches deep, is filled. As this volume is insufficient for a deep charge in the vat, a new quantity of extra-solution is made as soon as the first one has been allowed to sink to the surface of the ore, etc., until the extra-solution has displaced all the liquid previously contained in the ore, when, generally, circulation is commenced. It seems to me this method does not possess advantages over the first one, except that it does away with a special tank for preparation of the solution. The weighing of chemicals for each of the several charges required to saturate the ore, is troublesome, especially if the empty space above the ore should be unusually shallow, and the vat be deep.

Percentage of Copper Sulphate Used.—The consumption of copper sulphate used in preparing standard extra-solution for raw, acid, and arsenical alkaline ores varies between 0.6 and 1.2 per cent. of the solution; that of extra-solution for alkaline ores, between 0.1 and 0.33 per cent.

Volume.—The volume of standard extra-solution used for raw, acid, and arsenical alkaline ores is just sufficient to saturate the charge. It varies, according to specific gravity and fineness of pulverizing, for raw ores between 6½ to 9 cubic feet; for acid and arsenical alkaline ores, between 10 to 14 cubic feet per ton of ore. These small quantities, as compared with the volume of ordinary solution, are entirely sufficient to accomplish the object in view.

For alkaline ores, the quantity of extra-solution is from 4 to 6 times the volume to saturate the charge.

Depth.—The quantity of extra-solution required can also be expressed in percentage of the depth of the wet charge, that is, its depth after lixiviation has commenced. It is from 36 to 42 per cent. for raw ores; from 34 to 46 per cent. for acid and arsenical alkaline ores; and from 150 to 300 per cent. for alkaline ores.

Circulation.—In using standard extra-solution on raw and acid ores, it is generally circulated by the Korting ejector. Thus, the

small volume of this solution is repeatedly brought in contact with all the ore-particles, producing the best effect obtainable. A solution in motion is a much better solvent than one at rest. Still, there are exceptions to this rule—as will be seen in § 1407—quite difficult to explain.

The extra-solution on alkaline ores is not circulated, but manipulated like ordinary solution.

Effect of one Pound Copper Sulphate.—From practical experience it appears that in treating acid ores, 1 pound copper sulphate, consumed in standard extra-solution, extracts—after the ordinary solution has done its work—from 1.68 to 1.81 ounces silver per ton of ore. With alkaline ores, the effect—as compared with the work of ordinary solution—is greater.

Temperature.—Extra-solution is either used cold or heated to 35° to 50° C., or 95° to 122° F.

§ 1204. *The Sodium Carbonate Solution.*

If pure soda, manufactured by the Solvay process, has been bought, this solution does not require purification. It is made by saturating a sodium hyposulphite stock-solution with soda. In case ordinary soda-ash is at hand, its solution must be purified with extra-solution, which is added, while stirring, until a black precipitate ceases to appear. (See § 221).

§ 1205. *The Sodium Sulphide Solution.*

In order to obtain sodium sulphide free from caustic soda, converting the latter completely into sulphide and hyposulphite, the following "modus operandi" must be strictly followed:

Manufacture.—The whole contents of a drum of caustic soda, 600 to 700 pounds, broken up into lumps which should not exceed 5 or 6 pounds in weight, are placed into the iron tank previously described. About 3 to 3¾ cubic feet of water are added provided dry steam of high pressure is supplied, but less water is used if the steam is not dry and of low pressure. Steam is now turned on, and the tank covered to avoid spattering of the lye. The dissolving of the caustic soda takes about three-quarters of an hour, and at the end of the operation sufficient steam should be admit-

ted to raise the temperature of the lye to not less than 100° C. or 212° F. The condensed steam increases the volume of the solution. Its total volume should finally be such that it measures as many cubic feet as were pounds of caustic soda charged divided by 62½. In other words, one cubic foot of lye should hold about 62½ pounds of caustic soda. Now, pulverized sulphur, two-thirds of the weight of the caustic soda, is added, a shovel full at a time, while stirring.

The sulphur, if in lumps, should be broken so that it will pass a screen with openings 1 inch square. The sulphur melts, the temperature of the mass rises, it boils and foams, swelling to several times of its original volume while the chemical reaction takes place. If the process is conducted according to the directions given above, the sulphur will dissolve rapidly and completely. Should the lye have been either much less concentrated or of lower temperature than stated above, the solution of the sulphur will not be complete, and the lye must be boiled with steam for three or four hours, in which case the resulting product is liable to contain free caustic soda. The sodium sulphide solution, thus obtained, is so concentrated that it would solidify upon cooling. It is diluted with hyposulphite stock-solution in preference to water, and then transferred to the storage-tanks.

Dilution.—If the dilution is carried only so far that the total volume of the solution measures as many cubic feet as have been used pounds of caustic soda divided by 20, the reagent may have the highest possible precipitating-coefficient for silver =230. This, however, is only the case if proper precautions have been used in the manufacture. That high precipitating-coefficients are only produced at the expense of sodium hyposulphite, has been fully shown in § 606. A solution of this concentration is liable to crystallize, and stop up the iron pipes leading from the storage-tanks to the precipitating-tanks. If dilution is carried to such an extent that the total volume of the solution measures as many cubic feet as have been used pounds of caustic soda divided by 6, the reagent may have the the lowest normal precipitating-coefficient for silver =156. In using this solution, sodium hyposulphite will not be lost. Should the reagent contain free caustic soda, due to improper manufacture, the precipitating-coefficient sinks below 156. The statistics of the mill will show whether it is more profit-

able to precipitate with sodium sulphide of high or low coefficient. If the loss in sodium hyposulphite is so great—owing to decomposition of this salt by copper sulphate in making the extra-solution—that even the use of sodium sulphide of lowest precipitating-coefficient makes it necessary to add a considerable quantity of sodium hyposulphite to the stock-solution, then the question should be considered, whether it is more profitable to buy the latter salt, or to produce it in the mill by oxidation of sodium sulphide, that is, by using an old, and not a freshly made solution. This subject has been fully discussed in § 603.

§ 1206. *The Calcium Sulphide Solution.*

Kuestel recommends using for one part of sulphur, $1\frac{1}{8}$ parts caustic lime of purest quality, or 3 parts of the latter if it is impure, and 10 parts water. The caustic lime is charged into an iron tank, then water is added, and finally the pulverized sulphur. The latter should be sufficiently fine to pass a No. 30 wire-screen. Steam is admitted through an iron pipe, and the mass is kept boiling for 4 or 6 hours, stirring it frequently. More water, if necessary, must be added to keep the charge sufficiently liquid. At the end of the operation, the solution is allowed to stand undisturbed for 12 hours, when it appears clear above the undissolved residue. It is now drawn off. Its concentration should be at least 6° B. In the residue will be found orange colored crystals, which are, according to Kuestel, CaS_2. They require about 400 parts of water for their solution. The quantity of these crystals increases, the shorter the time of boiling, and the less sulphur used in proportion to the lime. Hence, it is economy to use an excess of sulphur. In that case, the solution contains CaS_6, not CaS_4. The orange colored crystals, which Kuestal describes as CaS_2, are, according to Shoene, $3CaO,CaS_4+12aq$.

CHAPTER XIII.

The Charging and Discharging of the Lixiviation-Vats. —The Second Wash-Water.—General Rules for Handling the Lixiviation-Solution.

In this chapter I shall treat certain manipulations in the lixiviation-process which are the same for all classes of ores.

§ 1301. *Charging the Lixiviation-Vats with Ore.*

Charging by Cars.—The charging of the vats should be done by cars. Arrangements with hoppers above the vats, including elevators and conveyors, are not economical. With roasted ores, which have to be wetted down on the cooling-floor, they have proven a failure wherever tried. There is, in every case, a limit to the economy of replacing manual labor by machinery. The best arrangement is to run a car-track on the central line of the rows of vats, and have cars so constructed that they dump the ore on both long sides. This insures at once a good distribution of the ore in the vat.

Charge the Ore Dry.—The ore must be charged dry. If tailings are worked, they should first be passed through a dry-kiln, and then screened. Roasted ores, of course, contain a certain amount of moisture from wetting down on the cooling-floor, but this is not detrimental. The effect of charging ore wet will be described and criticised in Chapter XV., on trough-lixiviation.

Charge the Ore Cold.—Roasted ore should not be charged hot,—that is at a temperature of 65° to 120° C., or 149° to 248° F.,—but cold,—that is below 50° C., or 122° F.,—if it is desirable to reduce the amount of silver taken up by the first wash-water to a minimum. (See § 517.)

Weight of the Charge.—The weight of the charge should not be roughly estimated, but ascertained as accurately as possible. With dry raw ore this is effected easily enough; it is more difficult, how-

ever, with wetted down roasted ore, when the moisture enters into calculation. In the latter case, the samples taken from each car should be kept in a box with tight cover, so that moisture is not reduced by evaporation. To conduct lixiviation in a rational manner, reliable statistics must be kept. A correct basis should exist on which the "*apparent extraction*" of the silver, that is the percentage expected in sulphides, calculated from the values of the roasted ore and the tailings, can be compared with the "*actual extraction*," or the percentage of silver actually obtained in sulphides, including the silver extracted from the first wash-water, by a clean-up in the mill.

If a notable discrepancy between the two figures exists, it should be investigated. Loss in silver may occur in many ways: by leakage of solution; by inadvertence in running silver-bearing solution to waste; by wash-water; and by stealing of sulphides, which has to be guarded against especially in Mexico. The losses in silver before the ore enters the vats, do not concern us here.

Filling the Vats.—The vats are filled to such an extent that the top of the charge, after the solutions have been applied, is about 12 inches—rather more than less—below the upper rim of the vat, for reasons that will become apparent in the next paragraph.

Shrinkage of the Charge.—The ore, whether raw or roasted, shrinks as soon as lixiviation-solution is applied to raw ores, or the first wash-water to roasted ores. This shrinkage is from 10 to 18 per cent. for raw, and from 12 to 24 per cent. for roasted ores. It is greater in the latter case on account of the removal of soluble salts. A few trials will establish how many tons a vat will hold to leave the required empty space above the ore, after lixiviation has commenced.

Precautions.—After the vat is filled, the charge is levelled with a hoe. In doing this, the laborer should not step into the vat, or forcibly press down the ore. Such irregularities interfere with the uniform filtering of the solutions through the charge.

§ 1302. *General Rules for Handling the Lixiviation-Solutions.*

Before entering into a detailed description of the modifications in lixiviation necessary in special cases, it is best to establish general rules that must be followed under all circumstances, and without which the best results, and the greatest economy, cannot be reached.

Volume to Saturate.—The first step to be taken is to ascertain how many cubic feet of solution are required to saturate one ton of ore. In § 1203, it has already been noted between which limits this quantity varies for different classes of ore.

First Wash-Water.—All roasted ores are first treated with water, called the first wash-water. Whether the first wash-water is followed by ordinary or by extra-solution, it is imperative, at the end of this operation, to stop the discharge of the liquor as soon as the water has sunk to the surface of the ore. The space in the vat above the ore, say 12 inches deep, is now filled with solution, and the outlet kept open until the solution has sunk to the surface of the ore. This operation is repeated until all the wash-water, which runs to waste, is pressed out, and displaced by lixiviation-solution.

Lixiviation.—Now, the discharge of the liquor is turned into the precipitating-tank. It is, of course, necessary to observe carefully when this change takes place, so that hyposulphite solution is not lost. Samples of the liquor must be tested frequently with sodium sulphide. Professors of metallurgy praise in their books, in glowing terms, the sweetness of an argentic hyposulphite solution, and recommend imbibing it freely as a substitute for other reactions. The men in the mill, for reasons of their own, do not take kindly to this drink! In fact, only more concentrated hyposulphite solutions, saturated with silver chloride, have a sweetish taste.

Extra-Solution.—In applying standard extra-solution to roasted or raw ore, after treatment with ordinary solution, and in the minimum quantity, namely, just sufficient to saturate the charge, the same *modus operandi* as described above is adopted, in order to displace the ordinary solution. As soon as this is accomplished,

the discharge of the solution from the lixiviation-vat is stopped, and circulation of the standard extra-solution is commenced, or it is left standing on the charge.

Second Wash-Water.—At the end of lixiviating with hyposulphite solutions, the second wash-water follows. Here again the discharge of the liquor is stopped as soon as it sinks to the surface of the ore. Several charges of water are now turned on, sufficient to displace the lixiviation-solution, when discharging into the precipitating-tanks is stopped, and the wash-water is allowed to drain off, in case the tailings are shovelled out, or remains, if the vat is discharged by sluicing.

Measurement of all Solutions.—But outside of these measurements which are absolutely necessary, it is recommended to measure in the same manner, all the wash-water and lixiviation-solution consumed, so that only a fixed quantity, established by experience, is passed through the charge. This method is somewhat tedious, on account of the frequent interruption of the process, but it is the only one to be recommended.

It can readily be seen that turning on solutions at random would only create confusion. It is not feasible to measure volumes either in the storage-tanks, or in the precipitating-tanks, since the former supply a number of vats, while the precipitating-tanks collect solution from several sources simultaneously. To use the last mentioned system of measurement, it would be necessary to have for each vat separate storage- and precipitating-tanks, which is not practicable.

The Rate of Lixiviation.—The rate of lixiviation is the number of inches in depth of a liquid which will pass through a charge of ore in a vat per hour. In other words, it is the facility with which a solution filters or sinks through a charge. It is important to know upon what circumstances the rate of lixiviation depends, especially since metallurgical literature is full of erroneous statements on this subject. We have to consider the depth of the charge, the fineness of crushing, and the character of the ore.

Influence of Depth.—The depth of the charge, in contradiction to old notions, does not influence the rate of lixiviation, because the slight increase of compactness at the bottom of the vat, due to increased pressure, is sometimes more than balanced by the greater "*head*" of the column of solution.

For instance, at Cusi was one large vat into which the roasted ore was charged to the depth of nearly 6 feet, while in all the other vats the depth of charge was only 22 to 24 inches. Yet the rate of lixiviation was greater for the 6-foot charge. Also, in the treatment of the raw Bremen tailings, the rate of lixiviation for charges 50 inches in depth, was the same as for only 24 inches.

Influence of Fine Crushing.—For raw ores, the rate of lixiviation diminishes with the fineness of crushing, and the presence of clay is very detrimental. Special difficulties are experienced in treating tailings from raw amalgamation, if they have been ground repeatedly in pans. I cite the Bremen tailings, Silver City, New Mexico, as an example.

The average fineness of that part which could be successfully treated was such that 87.8 per cent. would pass through a screen with 150 holes to the linear inch, or 22.500 holes to the square inch. With this fineness of material, a vacuum of 14 inches of mercury produced a lixiviation-rate of about one-half inch per hour, measured on the surface of the charge. For roasted ores, the fineness of crushing has no appreciable effect on the rate of lixiviation, if the dust is well chloridized, and charged dry into the vats. For instance, when treating Ontario ore, a charge of the finest flue-dust from the last dust-chamber of the Stetefeldt furnace lixiviated just as rapidly as a charge of the coarsest material from the furnace-shaft. This is due to the perfectly roasted condition of the flue-dust, which causes the charge to maintain a soft and half-floating condition during the whole of the process, unless it is allowed to drain and settle for a considerable time. On the other hand, at Lake Valley, with a Howell furnace without an auxiliary fire, the flue-dust was so poorly roasted that even when mixed with the coarsest ore, it proved a great hindrance to rapid filtration.

For ores roasted in a Stetefeldt furnace, there is practically no limit to the fineness of crushing allowable.

TABLE No. 1302.

SHOWING THE RATE OF LIXIVIATION.

Description of Material Treated.	Tailings or Ore.	Name and Location of Mine or Mill.	Rate of Lixiviation. Inches per hour.	Value of Material. Oz. Silver per ton.	Size of Screen Used.
Previously treated by other processes one to three times.	Tailings.	Bremen, Silver City, New Mexico,	½	11.5	Not Recrushed. Orig. size not known.
	"	Raymond & Ely & Meadow Valley, Pioche, Nev.		8.	
	Ore.	Sierra Grande, Lake Valley, New Mexico	8	12.	16 Mesh.
Chloridized material with alkaline first wash-water.	"	San Miguel, Cusihuiriachic, Mexico	5	50.	12 "
	"	Yedras, Sinaloa, Mexico	7	62.	26 "
	Tailings.	Sierra Grande, Lake Valley, New Mexico	8	12.	16 "
	Ore.	Ontario, Park City, Utah, 1883-4, (Experimental Plant)	10	88.	16 "
Chloridized material with acid first wash-water.	"	Sombrerete, Zacatecas, Mexico	12	48.	8 "
	"	San Antonio, Cusihuiriachic, Mexico	10	63.	12 "
	"	San Miguel, Cusihuriachic, Mexico	7	45.	12 "
	"	San Bartolo, Cusihuiriachic, Mexico	8	40.	12 "

NOTE.—All the above rates of lixiviation except for the Bremen tailings at Silver City are without vacuum below the filter of the vat.

Increasing the Rate of Lixiviation by the Korting Ejector.—The rate of lixiviation can be greatly increased by creating a vacuum below the filter of the lixiviation-vat. This is most conveniently effected by the Korting ejector, first proposed by the author for this purpose in lixiviation. It is applicable in all cases, except with such extremely fine material as the Bremen tailings. If the rate of lixiviation is, without a vacuum, from 6 to 8 inches, the Korting ejector will generally double it; if only ¼ to ½ of an inch, the ejector will often raise it to 6 or 8 inches.

Irregular Shrinkage of the Charge.—It occurs during lixiviation that, in consequence of irregular shrinkage, cracks form, either through the body of the ore, or the latter draws away from the sides of the vat. These defects should be at once corrected, especially in lixiviating with standard extra-solution. Otherwise, the solutions, following the way of least resistance, will do their work very ineffectually.

§ 1303. *The Second Wash-Water.*

After the extraction of the silver has been completed, the lixiviation-solution remaining in the ore must be displaced by the second wash-water. This is done by running a measured number of inches of water on the charge, sufficient to saturate the latter. The liquor is discharged into the precipitating-tanks until displacement has taken place. The second wash-water is allowed to drain off, and run to waste, if the vat is shovelled out, or it remains in the charge, if the tailings are sluiced, as already stated before.

The modus operandi of displacing the lixiviation-solution by the second wash-water, is the same as in displacing the first wash-water by lixiviation-solution. Other methods of applying the second wash-water are described by Mr. Daggett as follows: If, in treating raw ores, water is scarce and must be economically used, the lixiviation-solution is allowed to drain out of the charge, and is then followed by water to the extent of about 5½ cubic feet per ton of ore, all of which is allowed to pass into the stock-solution (that is, the precipitating-tanks). This method, it seems to me, does not answer the purpose as well as the one first described. If water is scarce, why not collect the second wash-water and use it again in place of allowing it to run to waste, after the lixiviation-solution has been displaced?

In regard to roasted ores, Mr. Daggett says:

"The mode of using the second wash-water for roasted ores is the same as for raw ores, if the descending first wash-water was followed by solution at the time the surface of the first wash-water reached the surface of the ore. But if the first wash-water has been drained from the charge* before the solution is used, then likewise the solution must be drained from the charge before the second wash-water is used. In this case the second wash-water is then applied by suddenly flooding the surface of the charge with about one-half inch of water, which descends through the charge and is allowed to pass into the stock-solution."

How a lixiviation-vat of 16 feet diameter is to be suddenly flooded with about one-half inch of water, Mr. Daggett does not

* A description of this method of draining the first wash-water will be found in § 1401.

state. For a charge 6 feet deep, one-half inch of water would be only about one-half cubic foot of water per ton of ore. In using so small a volume of water, the larger portion of lixiviation-solution, remaining in the drained charge, must necessarily be lost; besides, this statement does not agree with the figures in Table No. 1712, where the smallest volume of second wash-water used is given at 2½ cubic feet per ton of ore.

§ 1304. *Discharging of the Tailings and their Sampling.*

The removal of tailings by sluicing is by far the cheaper method, and requires from 12 to 18 cubic feet of water per ton of ore. If water is not abundant, that part of the first wash-water which runs to waste can be collected and pumped into a tank especially provided for that purpose. The sluicing is done with a hose, after opening the sluice-gate. Where sufficient natural pressure is not available, a Knowles fire-pump is used for sluicing. Sluicing permits the use of deep charges, which are rather inconvenient for shovelling.

Sampling the Tailings.—In sampling the tailings, according to Mr. Daggett's experience at Cusi, the six inches next to the filter sometimes show a less thorough extraction of the silver than the average of the whole charge, for reasons I can not explain. Hence, in starting new works, this subject should be investigated.

Silver Sulphide on the Surface of the Charge.—It may occur that the stock-solution contains extremely fine particles of silver sulphide in suspension, due to imperfect settling of the precipitates. In this case, the silver sulphide is arrested in a stratum on top of the charge. This portion of the tailings may be of higher value in silver than the original ore. It is carefully removed with a hoe. In some mills it is customary to return these tailings to the roasting furnaces. Much cheaper and simpler is the method of treatment with extra-solution which extracts the silver rapidly and completely. A special vat, provided for this purpose, may be used, or these tailings may be transferred to one of the vats before the charge is treated with extra-solution. If, in lixiviation, extra-solution was used last, these rich tailings could not have been formed. But, as will be seen shortly, the process is always ended with ordinary, and not with extra-solution. Tailings of

this character are met with in lixiviation-mills so frequently, that the question may be considered, if it would not be advisable to run all the lixiviation-solution, after precipitation, through a filter-press,—which could be done almost entirely by gravity—thus obtaining a stock-solution entirely free from silver sulphide?

CORRECTION.

On page 152, second line from top, it should read: "only about two-tenths of one cubic foot of water," in place of "one-half cubic foot."

CHAPTER XIV.

The Treatment of Roasted and Raw Ores.

All roasted ores require leaching with water, prior to the application of lixiviation-solutions, for reasons pointed out in § 502.

§ 1401. *The First Wash-Water.*

Leaching from Above.—After the surface of the ore in the vat has been levelled, water is turned on, either on top of the charge, or from below the filter. The former method is used, either if the ore is of such a character that little silver is extracted by the first wash-water, or if the extraction of a considerable amount of this metal is not objectionable. In admitting water from above, the outlet of the vat remains closed until the latter is full. The outlet is then opened, and leaching continued until the soluble salts are removed, as indicated by testing the liquor with sodium sulphide.

Leaching from Below.—Water is introduced from below the filter, either if the amount of silver extracted is to be diminished as much as possible, or if water is scarce, this method causing a saving of 25 to 40 per cent. wash-water, compared with the operation of leaching from above. The water is allowed to rise slowly through the charge until the tank is filled. Leaching from below is then stopped, the outlet of the vat opened, and as soon as the liquor sinks to the surface of the ore, leaching from above is commenced in the manner described before. This method requires some further explanation. In § 516, it has been shown that a concentrated leaching-brine, saturated with silver chloride and salts of the base metals, if diluted, becomes turbid, and parts with these salts the more completely the further dilution is carried. If this dilution takes place in the lixiviation-vat itself, the precipitated salts are arrested in the charge as soon as the current is reversed, and leaching from the top commences. Hence, the first wash-water will hold less silver in solution, but also remove a smaller amount of the base metals. The final effect is that the

precipitated sulphides are of lower grade in silver, and more sodium sulphide is consumed in precipitation. Hence, it is doubtful whether this practice is profitable or not, unless water must be used as sparingly as possible. In recovering silver from wash-water, by precipitating it with scrap-iron, the expense is not perceptibly increased by the presence of a slightly larger quantity of the precious metal.

Quantity of First Wash-Water.—The quantity of first wash-water consumed is, according to Mr. Daggett's paper, from 12 to 53 cubic feet per ton of ore. An exceptional case is Ontario ore, which consumed 138½ cubic feet. In Table No. 1712, detailed statements on this subject will be found. The lowest consumption of water, in the examples given, is, however, 31½ cubic feet per ton of ore.

Displacing the First Wash-Water.—In § 1302, I have described how the first wash-water, after it has completed its work, is displaced by lixiviation-solution. Mr. Daggett favors another method. In place of following the water with lixiviation-solution as soon as the surface of the ore becomes uncovered, it is allowed to drain off. Then, the outlet of the vat is closed, and lixiviation-solution turned on until the vat is full. Now, the outlet is opened, the discharge turned into the precipitating-tanks, and lixiviation actually commences. The principal advantage of this method is that the tedious watching for the appearance of the first silver-bearing solution is avoided, as well as the risk of losing solution by inadvertence. Mr. Daggett also claims that the stock-solution becomes less diluted. The correctness of this statement seems to me very doubtful, and must remain so without figures from actual observation. The most objectionable feature of draining the wash-water is the effect it may have on the mechanical condition of the charge in the vat, in settling and packing it, especially if the ore contains much fine material which should be kept in a floating condition. The displacement method, both for the first and second wash-water, is, no doubt, the best.

Leaching with Cold or Hot Water.—It has already been mentioned that roasted ores should be charged cold (see § 1301). This also implies that the first wash-water should be used cold. How materially the solubility of silver chloride is increased by heating a lixiviation-brine, has been shown in § 517.

Table No. 1401. a. records results of leaching on a large scale at Cusihuiriachic.

TABLE No. 1401. a.

SILVER DISSOLVED FROM COLD AND HOT ROASTED ORES BY THE FIRST WASH-WATER INTRODUCED FROM ABOVE.

No. of Charges Experimented on.	Value of Ore in oz. Silver Per Ton.	Per cent. of Salts Soluble in Water.	Calculated Value in oz. Silver Per Ton.	Value after Leaching in Mill. Oz. Silver Per Ton.	Ounces extracted by First Wash-Water.	Per cent. extracted by First Wash-Water.
HOT DRY ORE.						
5	34.6	14.8	40.6	28.3	12.3	30.3
33	34.2	14.1	38.8	27.6	11.2	28.8
COLD DRY ORE.						
19	37.9	15.0	44.6	39.4	5.2	11.6

Upon further examination, it was found, however, that all this silver did not actually leave the lixiviation-vat with the wash-water. More or less water, free from salts, exists under the filter of the vat, left there from the second wash-water of a previous charge. If the concentrated brine meets this water, silver chloride is precipitated under the filter, and subsequently again dissolved by hyposulphite solution. For Cusi ore, even when 25 to 30 per cent. of the silver in the ore had been dissolved and extracted from the ore by the first wash-water, only 3 to 9 per cent. actually left the lixiviation-vat, charging the ore hot; while 2 to 6 per cent. left the vat when the ore was charged cold. Consequently, the difference between the results of hot and cold charging is not practically as great as would appear from Table No. 1401. a.

Distribution of Silver in Wash-Water.—Table No. 1401. b. contains an interesting statement as to the distribution of silver in the first wash-water, which was applied from above. It shows that at least 90 per cent. of the dissolved silver were contained in the concentrated brine leaving the vat during the first 15 minutes, after leaching had been commenced.

TABLE No. 1401. b.

DISTRIBUTION OF THE SILVER DISSOLVED IN THE FIRST WASH-WATER FROM CUSI ORE.

Number of Charges Experimented on.	Value of Charges in oz. Silver Per Ton.	Depth of Charges in Inches.	Per cent. of Salts Soluble in water.	Calculated Ore Value in oz. Silver Per Ton.	Value after Leaching in Mill.	Oz. Silver Extracted by First Wash-Water.	Silver Contained in 1000 c.c. at end of ¼ Hour.	Silver Contained in 1000 c.c. at end of One Hour.	Silver Contained in 1000 c.c. at end of Two Hours.	Silver Contained in 1000 c.c. at end of Three Hours.	Silver Contained in 1000 c.c. at end of Four Hours.	Silver Contained in 1000 c.c. at end of Five Hours.	Total time of Leaching.
1	29.2	24	14	33.9	25.2	8.7	.180	.0064	.0032	.0028	.0008	none	19 hrs.
1	29.0	24	14	33.7	26.5	7.2	.108	.0034	.0020	.0009	trace	"	15 "
*1	37.6	24	15	44.2	23.6	20.6	.270	.0060	.0018	.0008	.0008	.0004	16 "
*1	37.6	24	16	44.8	29.0	15.8	.266	.0400	.0008	trace	trace	none	18 "
1	39.6	24	15	46.6	37.4	9.2	.150	.0116	trace	"	none	"	15 "

NOTE.—The Silver in the Wash-Water is expressed in grammes per 1000 c.c.

* The quantities of silver extracted, in treating the charges marked with a star, are so much larger than the solubility of silver chloride in brine warrants, that I must hold Mr. Daggett responsible for the correctness of these figures.

Silver extracted by the First Wash-Water from various Ores.—Table No. 1401. c. from Mr. Daggett's paper, gives the silver extracted by wash-water from various ores. The quantities should have been given in ounces per ton of ore, and not in per cent. of its value.

TABLE No. 1401. c.

SILVER EXTRACTED FROM ROASTED ORES BY THE FIRST WASH-WATER, THE COST OF PRECIPITATING IT, AND THE VALUE OF THE PRECIPITATE.

Name of Mine.	Wash-Water Introduced from Above or Below.	Mode of Precipitating.	Per cent. of Ore-Value Obtained from Wash-Water.	Value of the Precipitate in Silver per ton.	Cost of Precipitating Per Ton of Ore.
Ontario (1883-84.)	Below.	Acid and Scrap-Iron.	0.85		
Lake Valley	Above.	" "	29.	9000 oz.	9 cts.
Sombrerete	"		Trace.		
San Bartolo	Below.	Sodium Sulphide.	3.3	1600 oz.	12 cts.
San Antonio	"	" "	6.4	1400 oz.	15 cts.
San Miguel	"	" "	7.8	2400 oz.	15 cts.
Yedras	"	" "	2.0	13934 oz.	6 cts.

The result does not depend upon the richness of the ore in silver, but—outside of temperature—altogether upon the concentration and composition of the leaching-brine. The theory of this subject has been fully discussed in Chapter V., and does not require further elucidation. If the first wash-water is carefully handled, and its silver extracted, it really makes little difference how much silver is dissolved in it.

At Lake Valley, for instance, according to Mr. Russell's statement, the loss from that source was only 0.07 ounces silver per ton of ore.

Treatment of the Silver-Bearing Wash-Water.

The extraction of silver can be effected by three methods.

§ 1402. *By Dilution with sufficient Cold Water.*

This method is not recommended. It requires large tank-capacity, the settling of the silver chloride being slow and imperfect, so that considerable silver is lost; the product is of low grade, salts of the base metals being precipitated as well as silver chloride. Since clear water is not everywhere and always obtainable, the mud and organic matter in the large quantity of water, necessary for dilution, will settle together with the silver-bearing precipitate deteriorating its value still further.

§ 1403. *By Precipitation with Sodium or Calcium Sulphide.*

This method recommends itself if the wash-water is alkaline in consequence of the presence of caustic lime in the roasted ore. Such wash-water, necessarily free from chlorides and sulphates of the base metals, and containing only a small amount of lead and zinc, besides antimony and arsenic, will yield a silver product of high grade, and requires a comparatively small quantity of the reagents for precipitation. Where sulphuric acid is expensive, and a considerable quantity would be required for imparting to the solution an acid reaction, prior to precipitation by scrap-iron, the method would no doubt be the cheapest. For execution of the operation, the ordinary precipitating-tanks with swinging pipe-discharge are used. That precipitation of the silver as a sulphide is not economical if the wash-water is neutral or acid, carrying considerable copper, zinc, manganese, and lead, is hardly necessary to point out.

§ 1404. *By Precipitation with Scrap-Iron.*

Most applicable is this method for wash-water containing copper, which is gained as a valuable by-product. The finely divided cement-copper greatly assists the precipitation of the silver. The process is best conducted in the apparatus and manner described below.

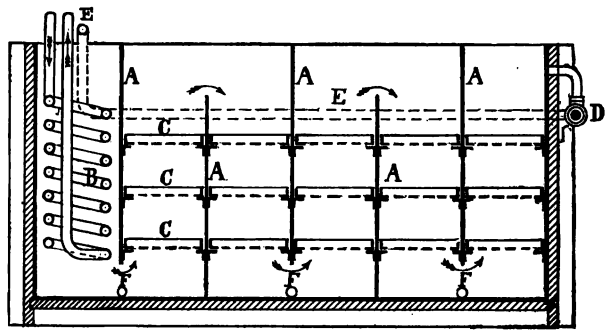


Fig. 28.

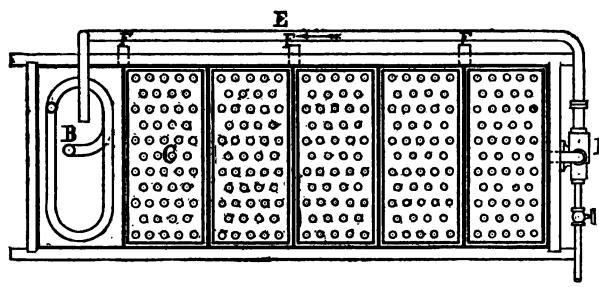


Fig. 29.
Wash-Water Precipitating-Tank.

The wash-water is collected in a lead-lined, oblong tank, provided with partitions, A., dividing the space into compartments communicating with each other alternately at bottom and top. A coil of lead-pipe, B., through which steam is passing, placed in the first compartment, heats the solution. The other compartments have movable, perforated trays, C., on which scrap-iron is placed. After the solution has been made acid, it is circulated by means of a Korting ejector, D., which draws the liquor from the top of the last compartment, returning it to the first compartment by the lead-pipe or rubber hose, E. The openings, F., serve to discharge

the solution, cement-silver, and cement-copper, after the precipitating of these metals is completed. Addition of sufficient sulphuric acid, and covering the tank, prevent the formation of basic ferric salts. The solution is allowed to settle in a tank with swinging pipe-discharge. Finally, cement-silver and copper are collected, after washing, in a filter-press. The best method of refining the precipitate is indicated in § 1606.

Although an apparatus like the one described has never been constructed and used, its utility is unquestionable. How quickly a charge of solution will be finished, an experimental plant, at small cost, will easily establish. The continuous circulation of the hot liquor will, undoubtedly, reduce the time of the operation to a minimum.

Table No. 1401. c., gives the values of precipitates from the first wash-water, and other information. As a rule, the precipitate from wash-water with alkaline reaction is higher in silver than that obtained from acid ores. Scrap-iron precipitated from the first wash-water of Ontario ore, (1883–84) 3¼ pounds copper per ton of ore.

The Lixiviation.

The mode of treatment with hyposulphite solutions varies according to the character of the ore, whether it is raw or roasted, and in the latter case, whether it is acid, arsenical alkaline, or alkaline.

§ 1405. *Lixiviation of Raw Ores.*

A standard extra-solution is generally used first, with [or without] circulation, followed by ordinary solution. In rare instances, the standard extra-solution is preceded by ordinary solution. In this, and in other instances, it is not always possible to give "the reason why." Cases exist where practical experience alone gives us a clue. Try different methods and select the best.

How standard extra-solution is prepared for raw ores, has already been described in § 1203.

Explanation of Table No. 1405. For other details in manipulating raw ores, namely, time of lixiviation,* temperature of solu-

* The time of lixiviation depends on the rate of lixiviation.

tion, etc., I refer the reader to Table No. 1405, copied from Mr. Dagget's paper. In it, "*cold*" means ordinary temperature, such as the Signal Service furnishes in a moderate climate, but not as during a Montana blizzard; "*hot*" is a temperature not above 50° C., or 122° F.; "*ordinary*" is the common stock-solution; a "*weak extra*" is an extra-solution containing more sodium hyposulphite than is required for standard composition, being, at the same time, low in copper; an "*extra*" is an extra-solution of standard composition; a "*special extra*" is a standard extra-solution which has already been used on a charge of ore, but still retains sufficient cuprous hyposulphite to do more work. Hence, it is, by the Korting ejector, transferred to another vat, and not discharged into the precipitating-tank. In this way considerable saving in chemicals is effected. This "*special extra*" is never circulated, but passed through the ore like an ordinary solution, and is then turned into the precipitating tanks. A "*charge*" is the volume of solution required to saturate the ore in the lixiviation-vat.

§ 1406. *Lixiviation of Acid Roasted Ores.*

Ordinary solution always precedes standard extra-solution—generally circulated—which is followed again by ordinary. Here I have to add some remarks about the volume of standard extra-solution. It has been stated, in § 1203, that this volume is taken just enough to saturate the charge. If an extra-solution follows an ordinary one, and is circulated, its volume is taken about 10 per cent. short of that required for saturation. It can readily be seen that it would be to no purpose, in displacing ordinary solution by standard extra-solution, to run a part of the latter into the precipitating-tanks before it has been circulated and done its work, which might occur if the volume were taken a trifle too large.

Effect of Circulation.—That circulation of standard extra-solution is very essential for acid roasted ores in producing maximum results, becomes evident from Table No. 1406. The same experience has been made with this class of ores in other mills.

TABLE No. 1405.

ORDER IN WHICH THE LIXIVIATION-SOLUTIONS MAY BE APPLIED IN THE TREATMENT OF RAW AND ROASTED ORES.

1		8-10 charges, cold ordinary..	1 charge, special extra.....	Cold ex. circulat'd 3-5 hrs.	3-5 charges, hot ordinary	Raw Ore and Acid Roasted Ore.				
2		6-8 charges, hot ordinary....	1 charge, special extra.....	Hot ex. circulat'd 2-4 hrs.	4-6 charges, hot ordinary (1st stands 4-6 hours)...					
3		8-10 charges, cold ordinary..	2-3 charges, hot ordinary....	Hot ex. circulat'd 2-4 hrs.	3-4 charges, hot ordinary					
4			10-12 charges, cold ordinary.	Cold ex. circulat'd 3-5 hrs.	3-5 charges, hot ordinary					
5			6-8 charges, hot ordinary....	Hot ex. circulat'd 2-4 hrs.	4-6 charges, hot ordinary (1st stands 4-6 hours)...					
6		6-8 charges, hot ordinary....	Hot extra circulated 2-4 hrs.	The same stands 6-8 hrs..	4-6 charges, hot ordinary (1st stands 4-6 hours)...					
7	8-10 charges, cold ordinary..	1 charge, special extra.....	Hot extra circulated 2-4 hrs.	The same stands 6-8 hrs..	3-5 charges, hot ordinary					
8			4-6 charges, weak cold extra	Strong cold extra stands 10-12 hours.....	10-12 charges, cold ord....		Alkaline Roasted Ore.			
9			1½ to 4 lbs. Copper Sulphate per ton in 1st Wash-Water	4-6 charges, weak cold ex.	12-14 charges, cold ord....					
10			1½ to 4 lbs. Copper Sulphate per ton in 1st Wash-Water	4-6 charges, weak cold ex.	10-12 charges, hot ord....					
11			All the Cop. Sul. in 1st W.W.	1 cold ord. stands 3-6 hrs..	12-14 charges, cold ord....					
12			All the Cop. Sul. in 1st W.W.	1 cold ord. stands 3-6 hrs..	10-12 charges, hot ord....					
13			All the Cop. Sul. in 1st W.W.	14-16 charges, cold ord....						
14		8-10 charges, cold ordinary..	3-4 charges, hot ordinary....	Hot ex. stands 10-12 hrs..	4-5 charges, hot ordinary		Alkaline Arsenical Roasted Ore.			
15		8-10 charges, cold ordinary..	1 charge, special extra.....	Cold ex. stands 10-12 hrs.	4-6 charges, hot ordinary					
16		8-10 charges, cold ordinary..	1 charge, special extra.....	Hot ex. stands 10-12 hrs..	3-5 charges, hot ordinary					
17		6-8 charges, hot ordinary....	1 charge, special extra.....	Hot ex. stands 10-12 hrs..	4-6 ch'g's, h. ord. (1st st'ds)					
18			6-8 charges, hot ordinary....	Hot ex. stands 10-12 hrs..	4-6 charges, hot ordinary (1st stands 4-6 hours)...					
19			8-10 charges, cold ordinary..	Cold ex. stands 12 hrs. ...	4-6 charges, hot ordinary					
20		1½ to 4 lbs. Copper Sulphate per ton in 1st Wash-Water	4-6 charges, weak cold extra	Strong hot ex. cir. 4-6 hrs.	10-12 charges, hot ordinary (1st stands 4-6 hrs.		Alkaline and Acid Mixed.			
21	1½ to 4 lbs. Copper Sulphate per ton in 1st Wash-Water	4-6 charges, weak cold extra	Cold extra circulated 3-5 hrs.	The same stands 6-8 hrs..	12-14 charges, cold ord....					
22		1½ to 4 lbs. Copper Sulphate per ton in 1st Wash-Water	4-6 charges, weak cold extra	Strong hot extra stands 10-12 hours.....	10-12 charges, hot ord....		Alkaline & Alkaline Arsenical Mixed.			
23	1½ to 4 lbs. Copper Sulphate per ton in 1st Wash-Water	4-6 charges, weak cold extra	2-4 charges, hot ordinary....	Hot ex. stands 10-12 hrs..	5-6 charges, hot ordinary					
24				Cold ex. circulat'd 4-6 hrs.	12-14 charges, cold ord....					
25				Hot ex. circulat'd 4-6 hrs.	10-12 charges, hot ord....					
26				Cold ex. stands 10-12 hrs..	12-14 charges, cold ord....					
27				Hot ex. stands 10-12 hrs..	10-12 charges, hot ordinary (1st stands 4-6 hrs).					
28			Hot extra circulated 4-6 hrs.	The same stands 10-12 hrs.	10-12 charges, hot ord....		Raw Ore.			

TABLE No. 1406.

EFFECT OF CIRCULATING THE EXTRA-SOLUTION AS COMPARED WITH NOT CIRCULATING, (ACID-ROASTED ORE) AT CUSI MILL, ON CUSI ORE.

"Circulated" or "Not Circulated."	Number of Charges Treated.	Value of Ore in Silver Per Ton.	Per cent. Extraction by Extra-Solution in Assay Office.	Per cent. Extraction in Mill.
The extra-solution stood in the ore for twelve hours.	46	34.8 ounces.	89.0	78.2
The extra-solution was circulated for four hours.	96	34.8 ounces.	87.6	85.1

§ 1407. *Lixiviation of Arsenical Alkaline Roasted Ores.*

Their treatment differs from that of acid roasted ores in the manipulation of the standard extra-solution. The latter is not circulated, but allowed to stand in the charge from 10 to 12 hours. It may then be circulated. This is a matter of experience, and I am not able to offer an explanation. That the difference in results between circulating and not circulating the standard extra-solution is considerable, is shown in Table No. 1407. It is interesting to compare this table with the preceding one.

TABLE No. 1407.

EFFECT OF ALLOWING THE EXTRA-SOLUTION TO STAND TWELVE HOURS IN THE ORE, AS COMPARED WITH CIRCULATING THE SAME (ALKALINE-ARSENICAL ROASTED ORE), AT YEDRAS MILL, ON YEDRAS ORE.

"Circulated" or "Allowed to Stand Twelve Hours."	Value of Ore in Silver Per Ton.	Per cent. Extraction by Extra-Solution in Assay Office.	Per cent. Extraction in Mill.
The extra-solution was circulated for four hours.	57.4 ounces.	83.3	77.7
The extra-solution stood in the ore for twelve hours.	63.5 ounces.	83.1	82.6

§ 1408. *Lixiviation of Alkaline Roasted Ores.*

Practical experience has demonstrated that if roasted ores, containing an appreciable percentage of caustic lime, are treated with ordinary solutions after the first wash-water, the result is very disastrous, and that the damage done is only partially made good by subsequent lixiviation with extra-solution. But if the extra-solu-

tion is applied at once, the cuprous hyposulphite neutralizes and counteracts the deleterious effect of the caustic lime, and leaves the silver in a soluble form. Before the extra-solution follows the first wash-water, it has been found beneficial to impregnate the ore with a small quantity of a strong copper sulphate solution. It can easily be seen that the first portion of the extra-solution, replacing the wash-water, must become more or less diluted by contact with water. This dilution may be so considerable that an insufficient quantity of copper is present to protect the silver from the deleterious influence of the caustic lime. The *modus operandi* is as follows: About one or two pounds of copper sulphate per ton of ore are dissolved in such a volume of water that it will fill the lixiviation-vat to a depth of two or three inches.

As soon as the first wash-water has sunk to the surface of the ore, leaching is suspended, and the copper solution is put on. The latter is made to sink a little below the surface of the ore before the extra-solution is run on the charge, and lixiviation is started. When the extra-solution has displaced the wash-water, the discharged liquor is turned into the precipitating-tank, and lixiviation with extra-solution is continued. The extra-solutions used for alkaline ores are neither of standard composition, nor are they circulated. They are operated like ordinary solution, and contain, as already stated in § 1203, an excess of sodium hyposulphite above standard composition, besides being of low concentration in copper. These extra-solutions are put on in several charges, commencing with one of $\frac{33}{100}$ per cent. copper sulphate, followed by others with less copper sulphate, the last one containing only $\frac{1}{10}$ per cent. of this salt.

I find in Mr. Daggett's paper a statement, referring to the treatment of alkaline roasted ores, which is difficult to understand. I quote him *verbatim:*

"But for some ores all the blue-stone may be used in the first wash-water. This may be done when the ore contains such a compound (usually a carbonate of an alkali or alkaline earth) as will precipitate all or a portion of the copper out of the wash-water, in the ore, in such a state that it will be wholly or partially dissolved by the ordinary or stock-solution afterwards used, thus making the extra-solution in the ore. The blue-stone, when thus used, is added to about one charge (or volume to saturate), of wash-water,

which may be either passed through the charge only once or circulated. If desirable, it may be followed by more wash-water, in order to wash out any copper not precipitated in the ore."

Whence a carbonate of an alkali in a roasted ore should come; or how a carbonate of an alkaline earth could exist after the ore has been red-hot, Mr. Daggett does not explain. A caustic alkaline earth, namely, caustic lime, is there sure enough. Supposing, however, that a carbonate of an alkali or alkaline earth existed in the ore, and precipitated cupric carbonate, how could the latter be dissolved by ordinary solution, and form an extra-solution, when cupric carbonate is entirely insoluble in aqueous sodium hyposulphite? The same is the case as to cupric hydroxide, precipitated by caustic lime. Why should it be desirable to wash out any copper not precipitated in the ore? Such vague statements are misleading. The practice of using all the copper sulphate in the first wash-water may produce the best results, but Mr. Daggett's chemistry is not correct. The following explanation seems to me more rational:

A solution of copper sulphate neutralizes the deleterious effect of caustic lime better than an extra-solution low in copper, and it is more effective and economical to neutralize caustic lime with a cupric than with a cuprous salt.

§ 1409. *Distribution of Silver and Lead in the Solution during different Periods of Lixiviation.*

Mr. Russell made experiments, at the Ontario Mill, to ascertain the distribution of silver and lead in the solution during the progress of lixiviation. He found that the first one-sixth of the solution, running out of the vat, contained 44.7 per cent. of the silver finally extracted, while the following portions decreased rapidly in their contents in silver. The last one-twelfth extracted only about 1 per cent. of the silver. The contents of the solution in lead, on the contrary, were nearly the same during the whole progress of the operation.

§ 1410. *Method of Determining whether the Extraction of the Silver is completed or not.*

If lead-bearing ores are treated, the facts recorded in § 1409 are of importance for two reasons. It is evident that lixiviation

should be suspended as soon as the silver is extracted, a continuation of the process furnishing but lead. The reaction with sodium sulphide can not be relied upon to determine whether the silver has been extracted from a charge or not, the lixiviation-solution giving a copious precipitate of lead sulphide. Even in the absence of lead, this reaction is not a safe guide, because the stock-solution, unless precipitation of the metals has been complete,—which should not be the case—always retains some silver and copper.

The only reliable way to ascertain whether the extraction of the silver is completed or not, is to assay the tailings, which should show a normal value, established by experience, nearly corresponding with the tailings obtained by assay-office lixiviation-tests. Very soon, after starting a mill, the volume of solutions necessary for best results becomes known, and investigation as to value of tailings becomes thereby simplified. That a change in character of the ore should be carefully watched, is hardly necessary to mention.

CHAPTER XV.

TROUGH-LIXIVIATION.*

Mr. O. Hofmann claims priority in having discovered that all the silver chloride in roasted ore is dissolved by hyposulphite solution, in ¾ to 1½ minutes, if the ore is introduced into a running stream of the solvent. It is rather quantity than concentration of the solution that produces this effect. The proportions of liquid and ore vary, and have to be determined by experiment in each case. Mr. Hofmann found, for instance, that this proportion for Cusihuiriachic ore was >12 to 1, and <18 to 1. To this method he has given the name of trough-lixiviation, because the solution is effected in triangular wooden troughs. The latter have a minimum length of 150 feet, with a fall of at least ¾ of an inch to the foot. Two separate plants are required, one for base-metal leaching, and one for silver lixiviation. The *modus operandi* is as follows: The ore is continuously charged into an agitator which breaks up lumps, and discharges into the base-metal leaching-trough. The latter ends above a system of settling-vats, connected with each other, and so arranged that any one vat can receive the charge from the trough, and can be disconnected without interrupting the operation of the whole system. Since the same arrangement is used in wet-crushing of silver-ores, a detailed description is not necessary. From the vat which is, at the time, the last one, the clear wash-water runs to waste. These vats are provided with filter-bottoms, and with central discharges for sluicing out the ore. As soon as a vat is filled, it is disconnected, allowing the pulp to flow into the next one. Fresh water is introduced upon the ore in the full vat to displace the base wash-water, and, finally, hyposulphite solution to displace the fresh water. The vat is now ready for sluicing with hyposulphite solution. The silver lixiviation-trough leads to a system of settling-vats of precisely the same arrangement and construction as the one described above, and the clear solution, charged with silver, runs

* Eng. and Mg. Journal, Sept. 10th and Nov. 26th, 1887.

into the precipitating-tanks. Whenever a lixiviation-vat is full, the hyposulphite solution is displaced by water, and the vat becomes ready for sluicing the tailings. Mr. Hofmann claims a great many advantages for this system, considering it superior in every respect to the ordinary practice. In doing so, he loses sight of the mechanical difficulties which this method presents. It can not be assumed that the number of vats will be lessened, as compared with vat-lixiviation, if the solutions are to be discharged perfectly clear. Such a volume of wash-water is obtained in the base-metal leaching, that it becomes impracticable to regain from it any silver it may hold. Mr. Hofmann states that no loss of silver will be incurred from this source. He bases his statement upon the fact that, according to Fresenius, silver dissolved in concentrated solutions of the chlorides of sodium, calcium, etc., is completely separated on sufficient dilution with cold water. Silver will, no doubt, go in solution when the ore is charged into the agitator. Even granting that this silver is again precipitated on sufficient dilution, it is questionable whether the fine particles of silver chloride will settle in the vats, or float off with the wash-water.

According to Mr. Chaney,* assayer at the North Mexican Mill, while Mr. Hofmann experimented with trough-lixiviation, the sulphides from two mill-runs of 450 and 300 tons, assayed 2953 ounces, and 1822 ounces silver per ton respectively, whereas sulphides produced by ordinary lixiviation from ore of the same character yielded on assay from 9506 ounces to 15877 ounces silver per ton. Mr. Chaney states that these low-grade sulphides were produced by precipitating the lixiviation-solution in a turbid condition, it being impracticable to allow sufficient time for a complete settling of the finest slimes before precipitation.

The greatest mechanical difficulty, however, will be met in the handling of the vats. It can readily be seen that as soon as a system of vats is in complete operation, each one will be filled with so many strata of ore of different grades of fineness as there are vats in the system, less one. The very finest slimes rest directly upon the filter, succeeded by strata of coarser and coarser material. The last stratum, which is formed when the ore enters the vat directly from the trough, is, of course, the largest one in

* Eng. and Mg. Journal, Dec. 3d, 1887.

thickness, all other strata taken together occupying much less depth. Besides this, the deposition of the ore on the filter is irregular. When the trough discharges the ore into the vat, the coarsest particles will drop to the bottom in one spot, thus making filtering in this particular place easier than elsewhere. Hence, in displacing, finally, the lixiviation-solution by wash-water, the latter will find its way through that portion of the charge, offering the least resistance, without permeating the finer strata. Thus, the pressing out of the lixiviation-solution may be apparently finished, while, in fact, a large portion of the solution is left in the ore, and is lost. Since filtering cannot be dispensed with in trough-lixiviation, this operation will not only become irregular, but very slow; in some cases almost impossible. Before Mr. Hofmann can show some decided success in practice, trough-lixiviation must be considered as one of the many processes that figure on paper only.

The fact that in lixiviation, not the solution of the silver, but the filtering takes time, was known long before Mr. Hofmann invented trough-lixiviation. Mr. E. H. Russell, while experimenting with the raw lixiviation of Bullionville tailings, wrote to me, on Jan. 3d, 1885, as follows: "In treating Bullionville tailings, we charge them in an agitator with muller revolving 28 times per minute. This agitator, which at the same time receives the hyposulphite solution, breaks up lumps, and discharges the tailings into a trough leading to the lixiviation-vats. *All the silver* (that can be extracted) *is practically dissolved by the time the tailings reach the vats*, but the difficulty is to get the solution through the filter, and, especially, to displace the solution by wash-water, etc. . . ."

The same observation was made at the Flagler Reduction-Works, Silver City, New Mexico. Others, however, had observed these facts before Mr. Russell. Hence, Mr. Hofmann cannot claim priority in having discovered the principle upon which trough-lixiviation is based. In the unfavorable view I take of trough-lixiviation, I am supported, so far as able to ascertain, by all who have had practical experience in lixiviation.

Mr. Daggett, in his paper on the Russell process, adds valuable criticism. He says:

"In the *Engineering and Mining Journal* of Nov. 26, 1887, the statement is made that for Cusi ore the minimum amount of ordinary or stock-solution required for trough-lixiviation was 12

parts of solution to 1 of ore, which is 384 cubic feet per ton, although the strength of the solution was 1.6 per cent. By referring to § 1202, it will be seen that the average amount of stock-solution used per ton of this ore, in the usual method of lixiviation, for a period of 9 months, and with a solution of 0.75 per cent. sodium hyposulphite, was only 80 cubic feet. Hence, trough-lixiviation, using a solution twice as strong, and requiring 4.8 times the volume, necessitated the use of 9.6 times as much sodium hyposulphite per ton of ore. One of the evil results of this would be the production of a lower grade of sulphides, and an excessive use of precipitant, as the amount of lead dissolved is directly proportional to the amount of sodium hyposulphite used. By referring to the account of some Yedras experiments in the same journal of Jan. 21, it will be seen that the injurious effect of caustic lime was the same while using trough-lixiviation as while lixiviating by the usual method, even though the tailings from the former were subjected to a subsequent treatment in the assay-office."

That the increased volume of stock-solution requires more labor in precipitation, and causes greater loss in sodium hyposulphite, is hardly necessary to point out.

After writing the above criticism, Mr. Hofmann's handsomely illustrated paper on Trough-Lixiviation, read before the Am. Inst. of M. E., at the Boston meeting, February, 1888, came to hand. I did not find anything in it to change my opinion.

CHAPTER XVI.

The Precipitation of the Metals from a Lixiviation-Solution.

§ 1601. *Constitution of a Lixiviation-Solution before Precipitation.*

The lixiviation-solution may hold, outside of silver and gold, the following elements that play an important part in precipitation, namely: Copper, lead, antimony, arsenic, and calcium. Neither iron, zinc, nor manganese can be present if the first wash-water has been properly applied.

Copper is derived from extra-solution, and from cuprous chloride in roasted ores; lead is introduced through lead sulphate in raw and roasted ores; antimony and arsenic are derived from antimonates and arsenates in roasted ores; calcium comes from calcium sulphate and caustic lime in roasted ores, and from gypsum in raw ores, it being difficult to remove an excess of calcium sulphate and hydrate entirely in leaching roasted ores. [It is here assumed that sodium sulphide is used as precipitant for the sulphides.] Although the quantity of the base metals in the lixiviation-solution rarely exceeds a few pounds per ton of ore, the value of the precipitated sulphides in precious metals may nevertheless be thereby diminished to a considerable extent.

The separation of silver, gold, copper, lead, antimony, and arsenic from the lixiviation-solution is effected by sodium [or calcium] sulphide, whereby, at the same time, sodium [or calcium] hyposulphite is regenerated. Calcium enters the list as one of the metals precipitated, if Na_2S_2 is used as precipitant, the sulphide CaS_2 not being easily soluble in water. This will be more fully considered in its proper place.

The only metal which can be separated by itself, prior to the precipitation of the precious metals, is lead. [Provided the solution is free from calcium-salts.] The precipitation of lead by sodium carbonate, and by caustic lime, has already been described

from a theoretical standpoint in § 221 and § 222. Here, details only of the practical execution of the method with sodium carbonate will be given.

§ 1602. *The Precipitation of Lead with Sodium Carbonate.*

The conditions required for this process have been fully set forth in § 221.

Manipulation.—The precipitation of lead is easily effected. While adding the soda solution, the contents of the tank are stirred with a wooden oar. The end of the reaction is best determined by taking a sample of the clear solution from the precipitating-tank in a test-tube, and adding a few drops of sodium phosphate, which gives a more voluminous lead precipitate, and a more delicate reaction than sodium carbonate. After complete precipitation of the lead, the solution is stirred for half a minute longer. The lead carbonate, being more compact than a precipitate of sulphides, settles quickly, and in half an hour the clear solution is decanted, and raised by an ejector into the silver precipitating-tanks. The lead carbonate is removed from the tank about once a week, and made into cakes by a filter-press. [Details of this operation are the same as for sulphides, described in § 1604.] It is very pure, unless calcium-salts were present in the solution before precipitation. The separation from silver is almost complete. At the Ontario, not more than 1½ ounces silver per ton were found in the lead carbonate. The only precaution this process requires is not to add an excess of sodium carbonate. While the presence of the latter in a hyposulphite solution is not at all injurious to the extraction of the silver, it can easily be seen that, should the stock-solution become caustic, after precipitation with sodium sulphide, it would be necessary to neutralize not only caustic soda, but also sodium carbonate.

Quantity of Lead dissolved in Lixiviation.—In § 225, it is stated that the solubility of lead sulphate—as such lead is principally contained in roasted ores—is materially decreased if a lixiviation-solution is of low concentration in sodium hyposulphite. A secondary influence has the temperature of the solution, a warm one being a better solvent than a cold one. Since the stock-solution rarely exceeds 1½ per cent. concentration in sodium hypo-

sulphite, it follows, that but a limited amount of lead is actually dissolved, even if the ore contains several per cent. of lead sulphate. In practice, the amount of lead extracted can be kept below 5 pounds per ton of ore, if proper precautions are used.

Economy of Precipitating Lead as Carbonate.—If an appreciable quantity of lead exists in a lixiviation-solution, it is always profitable to separate this metal by sodium carbonate. It is cheaper to precipitate lead as carbonate than as sulphide; the grade of the sulphides in precious metals is thereby raised, reducing expense of shipping, and charges for treatment, if they are sold to smelters; the treatment of the sulphides in the mill is facilitated, and lead carbonate is obtained as a valuable by-product. A calculation could easily be carried out,—assuming certain premises—covering all these items, but I shall confine myself to the first point.

The Precipitation of Lead by Sodium Carbonate and Sulphide Compared.—For precipitating 100 pounds lead, about 50 pounds pure Solvay soda are required according to equivalents, but somewhat more in practice, say 65 pounds.

The low, normal precipitating-coefficient of commercial caustic soda for lead is = 150; or 66.6 pounds caustic soda, consumed in the manufacture of sodium sulphide, precipitate 100 pounds lead. To this must be added, 44.3 pounds sulphur.

Cost of precipitating 100 pounds lead with sodium carbonate:

65 pounds Solvay soda, at 1.4 cents	
Freight, at 2 cents	$2.21

Cost of precipitating 100 pounds lead with sodium sulphide:

66.6	pounds	caustic soda, at 3 cents	$2.00
44.3	"	sulphur, at 1¼ cents	0.55
110.9	"	freight, at 2 cents	2.22
		Total	$4.77

Difference in favor of sodium carbonate: $2.56.*

Assuming that 5 pounds lead are precipitated per ton of ore, the saving is nearly 13 cents per ton. In running the experi-

* This calculation, however, needs a correction. In precipitating 100 pounds of lead with sodium sulphide, 59.8 pounds sodium hyposulphite would be added to the lixiviation-solution, representing a value, according to former calculations, of $2.39. On the other hand, we gain 100 pounds of lead as a by-product, the value of which, according to present quotations, would be $4.00, less freight and charges for smelting.

mental plant at the Ontario Mill, Mr. Russell found that by precipitating lead with soda-ash, the grade of the sulphides in silver was raised over 100 per cent., and their weight reduced 52 per cent. In selling sulphides to smelters, they generally pay for 99 per cent. of the silver, and charge about $100 for treatment. Hence, high grade sulphides leave a much larger profit to the mill.

§ 1603. *The Precipitation of Silver, Gold, and Copper.*

Manipulation.—The manipulation is the same, whether sodium or calcium sulphide is used; hence, I shall speak only of the former.

In adding sodium sulphide to the solution in the precipitating-tank, the contents of the latter are well stirred with a wooden oar, and this operation is continued for some time after precipitation is completed. The more violent the stirring, the quicker the settling of the sulphides, and the clearer the solution. In spite of this, it is often difficult to obtain a solution which, after decanting, is perfectly free from suspended sulphides. [The running of all the solution through a filter-press has already been proposed in § 1304.]

Testing for the End of the Reaction.—The end of the reaction is recognized if a sample of the clear solution in a test-tube gives only a slight precipitate with sodium sulphide. Of course, sodium sulphide must not be added in excess; a sample of the clear solution should not give a reaction with argentic hyposulphide.

Over-Precipitation.—If over-precipitation has taken place, more lixiviation-solution must be run into the tank; for this reason, the latter should not be completely filled before commencing precipitation. The addition of an excess of sodium sulphide is indicated by the odor of sulphuretted hydrogen. I can not say if this is invariably the case. Sulphuretted hydrogen may be formed if the solution contains acid salts; or it may be produced according to formula II., given in § 605. If, in the latter case, metals are absent which would be precipitated by sulphuretted hydrogen "in statu nascendi," this gas would escape.

It may occur that, by inadvertence, an over-precipitated solution is transferred to the storage-tanks; then, silver sulphide will be precipitated in the ore-charge, in running such a solution to one of

the vats. By immediate treatment with extra-solution, this silver is extracted—one of the advantages of the Russell process. Formerly, with ordinary lixiviation, such a charge had to be returned to the roasting furnaces.

Precipitation of Calcium Sulphide.—Should the lixiviation-solution contain calcium-salts, insoluble. calcium mono- or bi-sulphide may be precipitated by sodium sulphide, contaminating the precious sulphides to a great extent. The light-yellow calcium sulphide precipitate, however, does not appear until nearly all the silver, gold, and copper have fallen out. But it may obscure the reaction in testing the solution as to the precipitation of the precious metals. In this case, the following method is used. To a large test-tube full of solution, sodium carbonate is added in excess. This precipitates calcium carbonate, [also lead carbonate if lead has not been eliminated previously] but leaves silver, gold, and copper in solution. Now, to the latter, sodium sulphide is added. A precipitate shows that the separation of the precious metals is not yet completed.

Decanting the Solution.—After the sulphides have settled, the clear solution is decanted by the swinging pipe into the solution-sump.

The settling of the sulphides requires from 1 to 4 hours.

Value of Sulphides.—Table No. 1603, gives the values in silver and gold of sulphides produced by the Russell process in various mills.

TABLE No. 1603.

VALUE OF THE "SULPHIDES" AND THE COST OF PRECIPITATING THE SAME.

Name of Mine or Mill.	Ore or Tailings.	Sulphides "Roasted", or "Dried."	Ounces Silver per ton in the Sulphides.	Ounces Gold per ton in the Sulphides.	Cost of Precipitating per ton of Ore.
Leeds	Tailings.	Dried.	3000	Trace.	34 cts.
Bremen	"	Roasted.	8000	None.	14 "
Veta Grande	"	Roasted.	5400	20.	30 "
Silver Reef	Raw Ore.	Dried.	3300	Trace.	38 "
Raymond & Ely	"	Dried.	11600	13.2	20 "
Sierra Grand	Roasted Ore.	Roasted.	8000	None.	11 "
San Miguel	"	Dried.	8980	3.2	47 "
Yedras	"	Roasted.	15000	Not Assayed.	86 "
Veta Grande	"	Roasted.	8400	39.0	54 "
Sierra Grande	Roasted Tailings.	Roasted.	7200	None.	11 "
Ontario	Roasted Ore.	Dried.	10500	Not Assayed.	39 "
Sombrerete	"	Roasted.	7400	36.0	55 "
San Antonio	"	Dried.	8200	4.3	88 "
San Miguel	"	Dried.	7000	2.8	64 "
San Bartolo	"	Roasted.	8790	8.5	58 "

§ 1604. *Mechanical Handling of the Sulphides.*

The sulphides are removed from the precipitating-tanks about once a week. After decanting the clear solution as low as possible, the precipitate is discharged into the storage-tank for sulphides. Here, settling takes place again, and clear solution is decanted so as to save running it through the filter-press. The sulphides may be reduced in bulk by introducing steam into the storage-tank. If this is done, the mass should be allowed to cool before running it into the filter-press, because a warm solution destroys the filter-cloth much quicker than a cold one. The sulphides are now ready for the press.

Manipulation of the Pressure-Tank and the Filter-Press.—The valve between the storage-tank and the pressure-tank, as well as that between the latter and the filter-press, are opened, and the press is first filled by gravity. When the press is so full that solution runs out slowly, the valve between the storage and pressure-tanks is closed, and as soon as the liquid in the latter has sunk below the level of the small valve on its side, 3 or 4 inches below the top, steam* at full pressure is turned on (150 pound per square inch), until again the solution runs slowly from the filter-press, and the pressure-gauge indicates that complete filling has taken place. Now, the steam is turned off, the valve between the pressure-tank and the press is closed, and the latter discharged. For removing the cakes, a wooden and not an iron paddle should be used.

Capacity of Filter-Press.—A filter-press of 18 inches diameter, without distance-rings, has a capacity of about 250 pounds of dry sulphides per hour. Distance-rings increase capacity, but also moisture remaining in cakes.

Moisture in Sulphide-Cakes.—Sulphides obtained at 150 pounds pressure appear quite dry, but contain from 38 to 42 per cent. moisture.

Disadvantages of Leaving the Sulphides too long in the Precipitating-Tanks.—Mr. O. Hofmann condemns the practice of removing the sulphides only once a week. Freshly precipitated sulphides, he says, form large flakes which settle easily. After two or three days, they assume a dry and sandy condition, and, if

* It would be advisable to use compressed air in place of steam.

stirred up, divide into very fine particles which are kept suspended in the solution for a long time. The result of such a practice is that the decanted solution retains silver sulphide in suspension when returned to the storage-tanks, and silver sulphide covers the surface of the ore when the solution is used for lixiviation.

As a remedy, Mr. Hofmann recommends using small precipitating-tanks with machine-stirrers, and discharging the sulphides every other day. I consider the latter point well taken. The frequent occurrence of a stratum of rich tailings on the surface of the ore in lixiviation-vats, supports Mr. Hofmann's opinion.

§ 1605. *Drying of the Sulphides.*

It is of importance to remove from the cakes of sulphides, as they come out of the filter-press, every trace of moisture. This is best done in a chamber heated by steam-pipes, as illustrated in § 1111. Heating a drying-chamber by fire would easily ignite the sulphides, and spoil the sheet-iron pans on which the cakes rest. Sulphides are much easier sampled if dry than moist, and in roasting them, the mechanical loss is reduced to a minimum. If the sulphides are shipped to smelters, roasting is not necessary.

§ 1606. *Refining of the Sulphides.*

The Old Practice.—The old practice of treating the sulphides consists in roasting them in a reverberatory or muffle-furnace, and either melting the roasted material in black-lead crucibles with scrap-iron and borax, or in cupelling it with lead. The roasting is not carried to a dead roast, because by converting all the copper sulphide into oxide, fusion in a crucible would become difficult; hence, sufficient sulphur is left in the roasted material to form a matte. Scrap-iron is added to reduce, as much as possible, the contents of this matte in silver. If lead is present, it is, in part, precipitated by the iron, and assists materially in desilverizing the matte. In spite of this, however, a large percentage of the silver remains in the by-products. As an example I cite the experience at the Cusihuiriachic Mill, Mexico.

Crucible Melting at Cusihuiriachic.—Of the total silver in the roasted sulphides, according to Mr. Frank Johnson, crucible melting returned 81.8 per cent. in bars, and 18.2 per cent. in by-

products. The latter cannot always be reduced with profit in the mill, but are best sold to smelting-works.

Cupellation at Cusihuiriachic.—On the other hand, if the roasted sulphides are to be refined by cupellation, a dead roast is not desirable either. The cupelling process has the advantage of turning out refined bullion, but as to silver remaining in by-products, it is not much superior to crucible melting. At the Cusihuiriachic Mill, 83.1 per cent. of the silver were returned in fine bars, 14.2 per cent. remained in by-products, while over 2 per cent. were lost in cupellation. The cost of crucible melting was $1\frac{94}{100}$ cents, and of cupellation $2\frac{16}{100}$ cents per ounce of fine silver. To this must be added the cost of shipping the by-products to smelters, the smelting charges, and discount on the silver in the by-products. The more copper the sulphides contain in proportion to the silver, the greater will be the loss in by-products, and the expense of refining.

Loss of Silver in Roasting.—The loss of silver in roasting the sulphides in a reverberatory furnace may be very great. This loss is, in part, a mechanical one, but, principally, one by volatilization. Charging the furnace with moist sulphides increases the loss. Also the material of which the roasting furnace is constructed absorbs considerable silver, which, however, may not be reckoned a total loss. An examination of this subject at the Cusihuiriachic Mill, by Mr. Frank Johnson, showed that from 1 to 6 per cent. of the silver were lost in roasting, charging the sulphides dry; and as much as 12 per cent., charging them moist.

I consider this treatment of sulphides as a barbarous relic of the past, and wonder why metallurgists in charge of lixiviation-works, men of such pretentions as G. Kuestel and O. Hofmann, never made a progressive move, but clung tenaciously to methods that are out of date, and not in harmony with the enormous strides toward improvement characteristic of American metallurgical practice in every department.

Refining by Humid Processes.—It would be most desirable to avoid roasting the sulphides altogether, and with this object in view, I proposed, in 1884, a method of treating sulphides, as they come from the filter-press, by a humid-process. (See Trans. Am. Inst. of M. E., vol. xiii, page 47.) This process, however, which consists in its initial step in dissolving the sulphides in nitrated

sulphuric acid, has, so far, been executed only on paper. Hence, a decision regarding its merits must be deferred.

New Method Proposed.—Under the circumstances, I advocate the introduction of another method that is based upon a combination of well established operations. This method especially recommends itself to lixiviation-works using the Russell process, where sulphides are obtained free from lead, and composed, principally, of silver and copper. The dry sulphides are roasted in a muffle-furnace, the process being conducted to such a point that the sulphur is completely oxidized. At the same time a dead roast is not a desideratum. The more copper sulphate and silver sulphate remain, the better. The roasting gases, consisting, principally, of sulphurous acid, some sulphuric acid, nitrogen, and a surplus of air, are drawn through a Roessler* convertor, into which a previously roasted charge of sulphides has been placed. Before proceeding, I will describe the operation of the Roessler convertor.

The Roessler Convertor.—If sulphurous acid is passed through a hot, and not too acid solution of copper sulphate, it will rapidly change the blue color of this salt to a dirty green. Addition of cold water to the green liquor will precipitate metallic copper. This reaction proves that the sulphurous acid has reduced a portion of the cupric sulphate to cuprous sulphate, the latter remaining dissolved and undecomposed in a hot solution of copper sulphate. If, now, air is blown into the dirty green liquor, the blue color is restored, owing to the conversion of the cuprous sulphate into the cupric salt. At the same time the solution has gained in free sulphuric acid. If sulphurous acid and air are introduced simultaneously into a copper sulphate solution, the reactions of reduction and oxidation go on indefinitely with formation of free sulphuric acid. If the liquor reaches a certain concentration in free sulphuric acid, the reaction is weakened; hence, either cement-copper, scraps of metallic copper, or copper oxide should be present in the convertor to neutralize the free acid. If the sulphurous acid is not too much diluted with other gases, from 80 to 90 per cent. of it will be converted into sulphuric acid, while all the sulphuric acid, that may be present in the gases forced through the convertor, is also absorbed by the liquor. The acid

* Trans. Am. Inst. M. E., vol. xii, page 274.

generated in the convertor reaches a strength of 15° to 20° B.

A drawing of the Roessler convertor is shown, Fig. 30. A leaden tank, *B*, 8 or 9 feet high and 5 feet in diameter, carries a 6-inch lead pipe, *A*, branching at the bottom, and supporting the leaden ring, *C*, with numerous holes drilled on the lower side. A

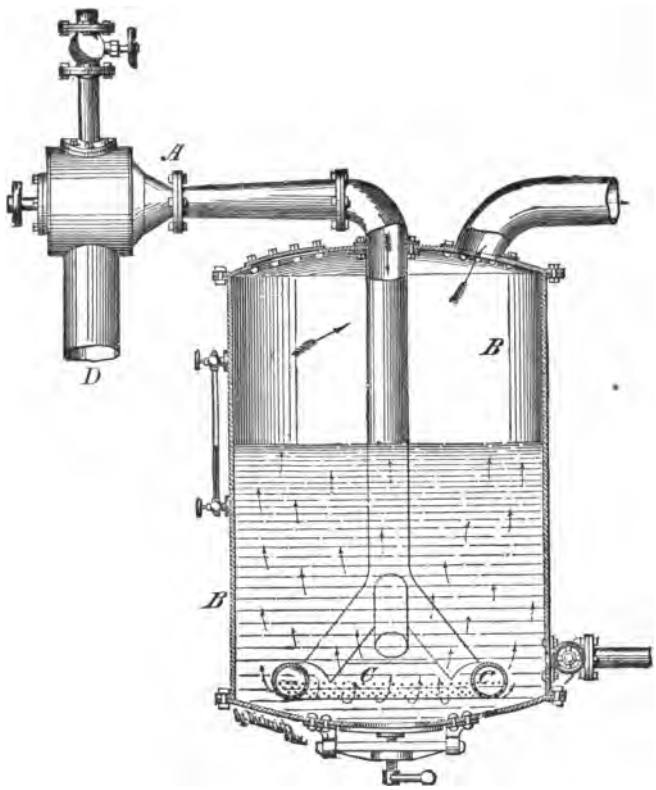


Fig. 30.—The Roessler Convertor.

Korting ejector draws the gases, containing the sulphurous acid and air, from the pipe, *D*, connected with the flue of the roasting furnace, and forces them through the copper sulphate solution which partly fills the convertor. This apparatus has been successfully introduced in many European refineries, where doré-bullion is parted by sulphuric acid, and where the condensation of the escaping sulphurous acid was a much desired object.

We now return to our process. Roasting in a muffle-furnace is essential, because if the roasting-gases were mixed with the pro-

ducts of combustion of the fuel, the sulphurous acid would be so much diluted that a large portion of it would escape conversion into sulphuric acid. At the end of the operation, the convertor will hold a solution of the copper oxide contained in the roasted sulphides, while all the silver will be found in the insoluble residue. The convertor is now discharged into a lead-lined tank, the solution, after clearing, is decanted, the residue washed, collected in a filter-press, and, finally, melted with borax into bars. From the copper solution copper sulphate is obtained by crystallization, while the acid mother-solution is returned to the convertor.

Advantages of the Process.—The great advantages of this method are:

1st. That it reduces the loss of silver in roasting to a minimum, all the fumes of volatilized silver being arrested in the convertor.

2d. Copper sulphate is gained as a by-product without the expense of sulphuric acid. The copper sulphate needed for the extra-solution is thus obtained without cost.

3d. Copper and silver precipitated from the first wash-water, by scrap-iron, can be refined in the convertor; and, if the wash-water contains much copper, the by-product of copper sulphate may be turned out in excess of the amount required for the extra-solution.

4th. More free sulphuric acid may be gained than is required for the solution of the copper, so that the acid mother-solution of copper sulphate may be utilized, in place of sulphuric acid, either for making acid the first wash-water, or for neutralizing a caustic hyposulphite solution.

It will be seen that none of the operations of this method present technical difficulties, or require expensive and complicated apparatus.

The antiquated treatment of the sulphides was one of the weakest, most objectionable points in the lixiviation-process, and a most unfavorable one in comparison with amalgamation.

§ 1607. *Selling Sulphides to Smelters.*

Although it is unquestionably more economical and satisfactory to refine the sulphides by a humid process at the mill, many lixiviation works prefer to sell their products to smelters, especially those of small capacity. The following information, furnished by

the manager of one of the largest lixiviation-mills in Mexico, will be useful.

The monthly shipments are divided into two parts, one being sent to the Chicago and Aurora Smelting and Refining Co., at Chicago, and the other to E. Balbach & Son, Newark, N. J. The division is made for the double purpose of having two checks on the work of the assayer at the mill, and also of having two refiners as checks upon each other. As the monthly shipments fill a number of packages, the division is made by sending the even numbered cases to one refiner, and the odd to the other, thus securing an average sample of the whole lot for each. The result of this method is, that the returns from both refiners are entirely satisfactory, the assays made at the mill and those by the buyers being virtually identical. The payments are made by check on New York, the silver being paid for on the basis of New York price on day of settlement, and gold at the rate of $20.60 per ounce.

E. Balbach & Son pay 99 per cent. of the value of the silver, full price for gold, and charge $100 per ton for treatment. The Aurora Works pay 98 per cent. of the value of the silver, and charge nothing for treatment. Their rates for gold are not known, as the sulphides in question contain only a trace. The Balbachs pay for the least trace of gold. The rate paid for copper is generally $1.00 per unit, although not in the case above mentioned, as very little copper is present.

The sulphides from the Sombrerete Mill are treated at the Argo Works, Denver, Col. The price paid by these refiners is 97 per cent. of the gold and silver (silver at N. Y. quotations and gold at $20 per oz.) without charge for treatment.

The sulphides from the Sombrerete Mill contain $700 to $800 in gold per ton. Taking into account the silver entering the cupel, the mill-assays for silver average about 0.6 per cent. above those of the refiners. Without taking this into account, the refiners' assays are a little higher than the mill assays. The average of the assays on gold agree within 0.1 per cent.

* It is hardly necessary to state, that these quotations are not fixed, but subjected to changes; they are merely given as an example.

CHAPTER XVII.

The Construction, Cost of Erecting, and Expense of Running a Lixiviation-Mill.

A. NOTES ON CONSTRUCTION.

§ 1701. *Capacity of Krom's Rolls and Stetefeldt Furnace.*

The most advantageous lixiviation-mill to build is one with two sets of Krom's 26-inch rolls, and one large Stetefeldt furnace. The capacity of the crushing-plant, supplemented by rock-breakers, is, of course, exceedingly variable, as it depends upon the fineness of crushing, and the character of the ore. Since ores, before chloridizing roasting, rarely require finer crushing than through a No. 30 wire-screen, the minimum capacity of two sets of 26-inch rolls may be safely set down at 40 tons per 24 hours. In many cases, however, with screening through No. 20 or No. 16 wire-cloth, the capacity of the rolls, if they are run to their maximum efficiency, may surpass that of a large Stetefeldt furnace. The capacity of the latter depends entirely upon the percentage of sulphurets in the ore. If the ore is so base that 70 or 80 per cent. is in the form of sulphurets, I consider 35 tons, put through one furnace in 24 hours, a maximum limit for really good work. Under such circumstances it may become necessary to erect two furnaces for two sets of rolls. In most cases, however, the ores contain only a moderate percentage of sulphurets, and then a large Stetefeldt furnace will easily handle from 60 to 80 tons of pulp in 24 hours, as has been demonstrated by actual experience. At the Ontario Mill, for instance, crushing in the battery through a No. 16 screen, the furnace was charged at the rate of 70 tons per day, and the extraction of the silver by the Russell process was above 90 per cent. The furnace at the Marsac Mill roasts 70 tons of pulp as thoroughly as when supplied with 50 tons. I am of the opinion, however, that 80 tons per day should be considered an economical limit of the capacity of a large Stetefeldt furnace.

§ 1702. *Arrangement of the Crushing-Plant.*

In arranging the crushing-machinery of the mill, I am in favor of having an independent building for the rock-breakers and ore-house, above the mill-building. The power to drive the rock-breakers can easily be transmitted by a wire-rope, and the broken ore, charged into cars from the shoots of the ore-house, brought by a tram-way to the drykilns. This arrangement has great constructive advantages. It requires less grade for the mill proper; allows the construction of a large ore- and salt-house; gives a better water-shed to the mill-roofs; more light and ventilation to the drykiln building, and lessens the danger of fire by not having the whole mill in one place, and under one roof. The ore-cars once charged, it makes little extra labor whether the distance to the drykilns is short, or about 200 feet long. This plan has been adopted at the Lexington Mill, Montana, where the ore-house is close to the mine, and the rock-breakers are driven by the engine supplying power to the machine-shop.

§ 1703. *Rock-Breakers.*

If the crushing is done by rolls, it is advisable to break the ore in the rock-breakers much finer—the largest pieces should pass a screen with holes 1 inch square—than is customary and profitable for stamps. Not only is the capacity of the rolls thereby materially increased, but the first set of rolls, receiving the ore from the drykilns, works and wears more regularly; besides, the drying of the ore takes less fuel, and if a shelf-drykiln is used, the ore is not liable to clog the narrow passages. Mr. Krom's rock-breakers are especially constructed for fine crushing, and they prepare the ore better for rolls than other machines of this class.

§ 1704. *Advantages of Perfect Drying.*

A perfect drying of the ore is essential for screening and subsequent mechanical handling of the pulp by elevators, conveyors, and the feeder of the Stetefeldt furnace. The ore should not only be charged into the first set of rolls perfectly dry, but warm, say at a temperature of about 100° C., or 212° F. Of course, the ore must not be excessively hot, or it would warp and even burn all

the wood-work of elevators, screen-frames, housings, and conveyor-troughs. Cold pulp lies dead and heavy; it mixes badly with salt; clogs shoots; wear out screens, and hangs at the mouth of a hopper, thus making it impossible to draw a stated quantity of pulp from the hopper by a screw-feeder. If cold pulp enters the Stetefeldt feeder its capacity is reduced, and the life of the screens shortened by excessive friction. Warm pulp, on the contrary, has more the properties of a liquid than of a solid—it runs. This is illustrated by the action of a charge of pulp in a reverberatory furnace after it has become heated, when the pulp flows and recedes if touched by a hoe.

§ 1705. *Details of Mill-Construction.*

Although it is not my intention to enter into all details of mill-construction, I shall elucidate some general principles which are often overlooked by inexperienced mill-wrights.

Feeders for Rolls.—The feeding of the first set of rolls with ore is best accomplished by a slowly revolving pulley, set below the mouth of the ore-hopper. Cone-pulleys regulate the speed of this feeder. The second set of rolls, to which all the coarse material from the screens returns, also receives the dry salt. The latter is fed from a hopper by a small screw-conveyor. If the rolls are to be used to their utmost capacity, a separate pulverizer, with screens, for salt must be provided, which, however, makes the mechanical arrangement of the crushing-room more complicated.

Elevators.—In constructing elevators, a rubber belt with sheet-iron or steel buckets is the cheapest and most convenient. If the elevator is inclined, Duck's rounded buckets can be used; if perpendicular, a bucket with square corners must be selected. Duck's bucket does not discharge well from a perpendicular elevator. The shoot feeding an elevator should enter on a level with the center of the bottom pulley, not lower down.

Shoots.—Place all shoots for coarse material at an inclination of not less than 45°; those for pulp, at an angle of not less than 60°. Do not contract shoots at the lower end. In placing shoots, avoid angles as much as possible; where an angle is necessary, give sufficient drop at the entrance of one shoot into another one.

Hoppers.—The sides of hoppers for pulp should have an inclination of 60°. Hoppers of 45° simply carry a superfluous dead weight of pulp.

Conveyors.—Screw-conveyors are best made by keying a row of cast-iron conveyor-flights, interlocking each other, to a wrought-iron tube. The length of a screw-conveyor should not exceed 14 feet. For longer distances, two or more sections have to be arranged, overlapping each other at the ends.

Pulp-Feeder.—The withdrawing of pulp from a hopper is done by a small screw-conveyor, driven by cone-pulleys. At least 3 or 4 flights of this conveyor should be exposed to the pulp. The sides of the hopper, near its bottom where the screw plays, must not be contracted to the outside diameter of the flights, but spread out at an angle of 60°, so that the sides form a tangent to a cylinder the diameter of which is from 1½ to 2 inches greater than the outside diameter of the screw-flights.

Screen-Tower.—There is only one screen-tower needed for two sets of rolls. It is located between them, and carries two concentric screens of Krom's pattern, one above the other. The inner screen is coarser than the outer one. Its object is to relieve the outer screen from unnecessary wear.

Most of these matters seem to be trivial, but I have seen the best mill-wrights make such gross mistakes in construction, from sheer ignorance of the properties of pulp, that these hints will be useful to the inexperienced.

Speed of Engine and Line-Shaft.—The engine in a mill, using rolls, should make from 120 to 140, and the main line-shaft, 200 revolutions per minute. The rolls run at a speed of 100 revolutions per minute. In giving to the engine and to the main line-shaft the speeds as stated above, small pulleys for transmission of power to the rolls, and a line-shaft of small diameter are the result.

Sellers' Hoist.—The Stetefeldt furnace is provided with a Sellers Hoist for convenience of the furnace fire-man, who has to attend to the feeder on the top floor; he also takes the pulp-samples.

B. Estimated Cost of Mill-Plant.

In the following specifications of mill-plant, I have selected what I consider best and most suitable for a lixiviation-mill. On many of the prices given, there is a discount, if payment is made in cash. Although these estimates are not complete in every respect, they will give a fair idea of the cost and weight of a plant, calculated to reduce about 80 tons of ore per day.

§ 1706. *Estimated Cost of Plant for Drying, Crushing, and Roasting the Ore.*

Description of Plant.	Cost.	Weight. Pounds.
*Buckeye Engine; style A.; cylinder 14-in. diam., 24-in. stroke; 130 revolutions per minute; 72 to 121 H. P.	$1775.00	15,000
†3 Hazelton Boilers, of 40 H. P. each	------	18,060
Knowles' Feed-Pump, No. 3	275.00	640
Heater	175.00	2,000
Air-Pump and Condenser	450.00	
2 Krom Rock-Breakers	1500.00	12,000
‡3 Double Shelf-Drykilns	2700.00	70,000
§2 Sets Krom's 26-inch Rolls	4500.00	29,000
2 Krom Screens	800.00	1,200
‖Stetefeldt Furnace	3000.00	49,000
Sellers' Hoist, with Cage and Merrick's Safety Catches	600.00	2,000
Shafting, Bearings, Pulleys, Wire-Rope Transmission	1300.00	18,000
Electric Light Plant, with separate Engine for Dynamo	1700.00	
Belting	1000.00	
Elevators, Conveyors, Feeders, Hoppers, Cone-Pulleys, etc.	1000.00	

* Buckeye Engine.

It has been and still is the custom to use in stamp-mills costly engines of low speed. For a crushing-plant with rolls, an engine of moderately high speed, from 120 to 140 revolutions per minute, is most suitable and economical. Of this class of engines, the Buckeye is one of the best in the market.

† Hazelton Boiler.

The Hazelton Boiler is not yet as well known as it deserves. It is beyond the scope of this treatise to enter into a description of this

boiler, and to point out all its advantages. After a visit to the manufacturing establishment of the Hazelton Boiler Co. in New York, where I examined all details of construction, and after seeing it in operation at the works of the New York Mutual Gas Light Co., I became convinced that the Hazelton is the best tubular boiler manufactured. Indeed, every claim set forth in the pamphlet of the company can be easily verified. I consider it especially well adapted to furnish dry steam of high pressure for the lixiviation-plant proper. Prices can be obtained by applying to the manufacturers. (See advertisement at the end of book.)

I draw attention to the light weight of the Hazelton boiler, as compared with other tubular boilers, whereby a saving of over 50 per cent. in freight is effected. A Hazelton boiler of 40 H. P. requires 5000 common, and 400 fire-brick for setting.

Of the three boilers, only two are actually needed, one being kept for emergencies. They should be so connected, that each two can be run independently of the third. The third boiler can also be used to supply steam for the lixiviation-plant proper, should the boiler in the lixiviation-building require cleaning or repairing.

The H. P. of engine and boilers is taken in excess of actual requirements, it being always good policy not to use either to their utmost capacity.

‡ Shelf-Drykiln.

One double shelf-drykiln requires about 30 thousand common brick, besides some stone-work, and about 400 fire-brick for lining the fire-places.

The shelf-drykiln is more economical in fuel, and wear and tear, than revolving dryers; besides, it does not require power, but acts by gravity. Another advantage is, that it does not produce dust like a revolving dryer, thus making dust-chambers superfluous. The ore from the shelf-drykiln is discharged only when needed to fill the hopper above the first set of rolls; hence, the rolls are always supplied with warm ore.

The iron plant for the shelf-drykiln is manufactured by Fraser & Chalmers, Chicago.

§ Krom's Rolls.

Although Krom's rolls are more costly than those furnished by other manufacturers, they are the cheapest in the end, and I consider them the most perfect and durable machine of this kind in the market.

‖ Stetefeldt Furnace.

A Stetefeldt furnace of largest size, including dust-chambers of ample capacity, chimney, and flue to the latter, takes from 300 to 350 thousand common, and from 8 to 10 thousand fire-brick, depending on the quality of common brick.

On their work, one mason with two helpers, will lay from 750 to 800 brick per shift of 10 hours. The lower part of furnace and dust-chambers, outside of foundations, requires about 3000 cubic feet of stone-work. One stone-mason and helper, will build about 5 perch of stone-wall in a 10 hour shift, provided not much dressing is needed. Correct patterns for the castings of the Stetefeldt furnace, revised by the inventor, are in the hands of Fraser & Chalmers, Chicago.

§ 1707. *Estimated Cost of the Lixiviation-Plant proper.*

The lixiviation-plant proper, as shown in the drawings, Plates I and II, would have, according to Mr. Russell's estimates, a capacity of 80 to 170 tons for various roasted and raw ores, if the tailings were removed by sluicing; and of 70 to 140 tons, if tailings were shovelled out.

Description of Plant.	Cost.	Weight. Pounds.
Lixiviation-Vats and Tanks: *		
Six Lixiviation-Vats.		
Staves and Bottoms, unfinished	$700.00	35,000
" " finished	915.00	
48 Hoops, 1⅛-in. round iron, 54 ft. long, at 3 cents per pound, unfinished	261.00	8,700
The same, finished	391.50	
Packing lumber in crates	91.00	
Three Storage-Tanks, and Six Precipitating-Tanks.		
Staves and Bottoms, unfinished	700.00	35,000
" " finished	915.00	
90 Hoops, ⅞-in. round iron, 35 ft. long, at 3 cents per pound, unfinished	193.50	6,450
The same, finished	290.25	
Packing lumber in grates	91.00	
One Sulphide Storage-Tank, One Solution-Sump.		
Staves and Bottoms, unfinished	152.00	7,500
" " finished	198.50	
14 Hoops, 1-in. round iron, 41 ft. long, at 3 cents per pound, unfinished	46.50	1,550

The same, finished	$69.75	
Packing lumber in crates	20.00	
ONE LOWER SOLUTION-SUMP.		
Staves and Bottoms, unfinished	26.25	1,300
" " finished	34.25	
5 Hoops, ⅞-in. round iron, 29 feet long, at 3 cents per pound, unfinished	9.30	310
The same, finished	13.95	
Packing lumber in grates	3.50	
Cast-iron and wrought-iron FIXTURES for VATS and TANKS, estimated	175.00	2,500
PIPES, VALVES, STEAM-HOSE	1,200.00	11,500
1 SODIUM SULPHIDE MIXING, and 2 STORAGE-TANKS	275.00	3,900
8 KORTING EJECTORS, No. 4	464.00	
KNOWLES' FIRE-PUMP, Size A, for sluicing tailings	425.00	1,600
KNOWLES' PLUNGER-PUMP, for pumping solution,	475.00	1,300
or KNOWLES' AIR-COMPRESSOR and PRESSURE-TANK (estimated)	750.00	3,000
JOHNSON'S FILTER-PRESS, 18 inches diameter	275.00	1,750
JOHNSON'S PRESSURE-TANK, for Sulphides	250.00	
HAZELTON BOILER, 40 H. P.		6,200
KNOWLES' FEED-PUMP, No. 2	200.00	

The plant for refining sulphides, by a humid process, as described in § 1606., would be as follows, viz:

MUFFLE-FURNACE with ROESSLER CONVERTOR.
JOHNSON'S FILTER-PRESS, 15 inches diameter.
Lead-lined cast-iron PAN for evaporating copper sulphate solution.
Several lead-lined TANKS for crystallizing copper sulphate.
KORTING EJECTOR for moving solutions.
REVERBERATORY FURNACE with gas-producer for melting silver; MOULDS for bars, etc.

The total cost of erecting such a refinery would not exceed $5,000.00.

* LIXIVIATION-VATS AND TANKS.

The total cost and weight of this part of the plant would be as follows, viz:

LUMBER.	HOOPS.
Unfinished: $1,578.25	Unfinished: $509.80
Finished: $2,062.75	Finished: $765.45
Weight: 78,800 pounds.	Weight: 17,010

Total weight: 95,810 pounds.

Total cost of finished tank-material, including packing: $3033.70

This estimate is from Fulda Brothers, San Francisco, Cal. It refers to dimensions given in the description of plant, Chapter XI. The lumber is best quality of Redwood or Oregon Pine, White Cedar not being in the market in sufficient quantity; the latter would cost 50 per cent. more. In the unfinished material, the two flat sides of the bottom-pieces will be planed; the staves will be planed both sides, and cut to length, but the edges will not be bevelled, only sawed straight, and not gained for inserting the bottom. The hoops will be straight bars of iron, without lugs.

The finished material will be ready in every detail, as described in Chapter XI, including the hoops.

All material, well packed, will be delivered, f. o. b. cars. Prices are cash on delivery.

Below is another estimate obtained from A. J. Corcoran, tank-buider, No. 76 John Str., New York. The lumber is White Pine, not entirely free from nuts.

		Pounds.
Staves and bottoms for 6 Lixiviation-Vats	$666.00	30,000
3 Storage-Tanks, and 6 Precipitating-Tanks	518.00	22,500
1 Solution-Sump, and 1 Sulphide Storage-Tank	146.00	7,500

Mr. Corcoran states that for finishing and setting up all the vats and tanks, the labor of 4 carpenters would be required for about 23 days. (10 hour shifts.)

"Painting Vats and Tanks."

Regarding the painting of vats and tanks, the following information will be useful.

Painting in white lead and oil, per 1000 square feet:

1st coat,	33.4 lbs. white lead;	2.17 gallons oil.
2d "	26.7 " "	1.34 " "
3d "	26.7 " "	1.34 " "
Total:	86.8 lbs. white lead;	4.85 gallons oil.

For the first coat, or priming, the thinner should be mostly oil. For second and third coat, very little turpentine should be used if the surface is to be glossy. One man will cover in a shift of 10 hours about 750 square feet with paint. (One coat.)

For all vats and tanks described, outside painting with three coats, would consume about 746 pounds white lead and $41\frac{7}{10}$ gallons oil. This work would occupy one man for 34 days.

§ 1708. *Estimated Cost of Erecting a Lixiviation-Mill.*

It is difficult to obtain reliable statistics; still more so to obtain a correct estimate regarding the total cost of erecting a mill. To give such an estimate in dollars would be misleading. It should be a detailed statement of the time of skilled and ordinary labor for stated quantities of material consumed in construction, as given in some of the marginal notes of § 1706 and 1707. At present, I am not able to furnish this information even approximately. I can only say, that the erection of a lixiviation-mill, as described here, is by far less costly than that of an amalgamating-mill of equal capacity. It is safe to assume that this difference is not less than 30 per cent.

C. Estimated Expense of Running a Lixiviation-Mill of 80 Tons Capacity Per Day.

§ 1709. *Labor.*

Where Used.	Number of Men.	Occupation.	Shift, Hours.
Engine and Boilers.	2.	Engineers.	12.
"	2.	Firemen.	12.
Ore-House.	2.	On Rock-Breakers.	10.
"	2.	Wheeling Ore.	10.
Drykilns.	4.	Bringing Ore from Ore-House, and Charging Drykilns.	12.
"	2.	Discharging.	12.
"	2.	Firemen, who also attend to Salt-Kiln, and assist man on Rolls.	12.
Rolls.	2.	Machinists.	12.
Stetefeldt Furnace.	3.	Firemen.	8.
Cooling-Floor.	12.	Discharging, Cooling, and Charging Lixiviation-Vats.	8.
Lixiviation.	2.	Lixiviators.	12.
"	3.	Helpers.	12.
"	2.	Precipitators.	12.
"	2.	Firemen on Boilers; Pumps and Pipes.	12.
"	2.	Handling Sulphides.	12.
Refinery.	2.	Roasters and Refiners.	10.
Mill at Large.	1.	Chief Metallurgist.	
"	½.	Chief Engineer, who is also Chief Engineer of the Mine.	
"	1.	Assayer.	
"	1.	Assayer's Helper.	

Mill at large.	1.	Bookkeeper.	
"	1.	Night-Foreman.	12.
"	1.	Machinist.	10.
"	1.	Carpenter.	10.
"	1.	Blacksmith.	10.
"	1.	Blacksmith's Helper.	10.
"	1.	Electrician.	
"	4.	General Help.	12.
"		Team and 2 Men Hauling Wood.	

The three helpers at the lixiviation-vats attend to the preparation of sodium sulphide, which occupies one man three hours once every two days, and to the preparation of extra-solution, which occupies one man about two hours, two or three times each day. Of the three helpers, one is on the night-shift, the other two on the day-shift, the sluicing of the tailings being done only on day-shift, and occupying one man about four or five hours each day. For tanks of 40 or 50 tons capacity, the two lixiviators and three helpers are more than sufficient, as the labor is much less than where tanks of ten tons capacity are used. If the plant is arranged as described, the pressing of the sulphides will occupy only one man three days per week. The removal of the dry sulphides from the steam-dryer, and recharging it with wet sulphides, occupies two men about four hours every three days, and the sampling, two men about six hours every three days.

As to the removal of tailings by shovelling, the cost at Cusi was 10 to 11 cents per ton of ore. At Sombrerete, the filling of the tanks and shovelling out the tailings, costs only 15 cents per ton, labor being cheaper there than that at Cusi. In the United States, with high-priced labor, the tailings should always be removed by sluicing.

§ 1710. *Fuel.**

	Cords Wood,	or	Tons Coal.
ENGINE BOILERS,	5 to 6.		2½ to 3.
BOILER, LIXIVIATION-BUILDING,	3		1½
DRYKILNS, Ore and Salt,	3½		1¾
STETEFELDT FURNACE, depending on quality of wood and character of the ore	6½ to 8½.		
SULPHIDE REFINERY, for roasting, refining, and melting bars in a reverberatory furnace	1		

* It is here assumed that one cord of wood is equal in effect to 1000 pounds of coal. This, of course, is not always the case. Estimates on fuel are, necessarily, approximate only, on account of great difference in quality.

§ 1711. *Wear and Tear.*

Thirty-five dollars per day would be ample to cover all ordinary breakage, and wear and tear of machinery, boilers, screens, dry-kilns, furnaces, lixiviation-plant, and refinery, including lubricants, electric light, and sundries.

At Yedras, the cost of labor and material for repairs on the lixiviation-plant proper was only $50.00 per month, or about 4½ to 5 cents per ton of ore. For a lixiviation-plant as described and shown in plan, running at a capacity of 80 tons per day, the total cost of labor and material for repairs would not exceed $150.00 per month.

§ 1712. *Chemicals.*

The consumption of chemicals, namely: Salt, sodium hyposulphite, copper sulphate, Solvay soda, caustic soda, sulphur, and sulphuric acid, has already been stated in previous chapters. A summary of these statements is found in Table No. 1712, which also contains the quantities of water consumed. The consumption of salt may vary between 60 and 240 pounds per ton, for ores that require chloridizing-roasting. Using extra-solution, it will rarely become necessary to consume such a large quantity of salt as 240 pounds per ton.

§ 1713. *Total Estimated Cost of Lixiviation.*

From the information given above, a calculation as to total cost of lixiviation per ton of ore can easily be made for a given locality. For plants of small capacity, the expense per ton of ore is, of course, materially higher so far as labor is concerned, but is not very much increased otherwise. Finally, fire-insurance, taxes, interest and amortisation on the capital invested in the mill, and its share of other expenses, namely: General management, legal expenses, and sundries, should be added wherever it is desirable to enter into close calculations as to profit and loss in working ores.

The following statistics, as to total expense of lixiviation, are copied from Mr. Daggett's paper.*

* The figures given refer to the mill-expenses proper, and do not include general expenses.

TABLE No. 1712.

AVERAGE AMOUNT OF CHEMICALS AND VOLUME OF WATER USED PER TON ON RAW AND ROASTED ORES AND TAILINGS TO WHICH THE RUSSELL PROCESS HAS BEEN APPLIED.

General Description of Material Treated.	Tailings. or Ore.	Name and Location of Mine or Mill.	Pounds of Copper Sulphate Per Ton.	Pounds of Sodium Hypo. Per Ton.	Pounds of Caustic Soda Per Ton.	Pounds of Sulphur Per Ton.	Pounds of Caustic Lime Per Ton.	Pounds of Sulphuric Acid Per Ton.	Cu. Ft. of First Wash-Water Per Ton.	Cu. Ft. of Second Wash-Water Per Ton.	Total Cu. Ft. of Water Per Ton.*
Previously Treated by Other Processes.	Tailings.	Leeds, etc., Silver Reef, Utah.	1.8	3.76	3.00	3.00	....	0.25	....	10.5	10.5
	"	Bremen, Silver City, N. Mex...	4.1	3.00	1.75	1.1	....	0.25	....	9.0	9.0
	"	R. & E. & M. V., Pioche, Nev..	2.0	1.50	1.50	1.5	....	....	....	9.5	9.5
	"	Veta Grande, Parral, Mexico.	4.0	2.75	....	4.0	5.0	0.25	....	11.5	11.5
Raw Ore.	Ore.	Raymond & Ely, Pioche, Nev.	2.0	1.50	1.50	1.5	....	0.25	....	9.5	9.5
Roasted Material with Alkaline First Wash-Water.	Ore.	Sierra Grande Lake Valley, New Mexico	2.6	2.6	1.4	0.9	....	1.80	39.5	2.5	42.0
	"	San Miguel, Chihuahua, Mex..	4.7	3.7	4.0	2.7	....	....	37.5	2.5	40.0
	"	Yedras, Sinaloa, Mexico	9.6	1.4	5.5	3.6	....	....	32.0	14.0	46.0
	"	Veta Grande, etc., Parral, Mexico	6.0	4.0	4.5	3.0	....	....	41.5	2.5	44.0
	Tailings.	Sierra Grande, Lake Valley, New Mexico	2.6	2.6	1.4	0.9	....	1.80	31.5	2.5	34.0
Roasted Material with Acid First Wash-Water.	Ore.	Ontario, Park City, Utah, 83-84	7.0	5.0	5.1	3.4	....	1.00	138.5	2.5	141.0
	"	Sombrerete, Zacatecas, Mex...	7.0	5.3	4.9	3.3	....	....	47.0	2.5	49.5
	"	San Antonio, Chihuahua, Mex.	8.3	7.0	7.75	5.0	....	....	52.5	2.5	55.0
	"	San Miguel, Chihuahua, Mex..	5.0	3.7	5.0	3.4	....	....	37.5	2.5	40.0
	"	San Bartolo, Chihuahua, Mex.	5.0	3.0	5.0	3.3	....	....	37.5	2.5	40.0

* This does not include the water required for removing tailings by sluicing.

Table No. 1804. a. gives the total mill-expenses per ton of roasted ore in the Cusi mill for nine months, while using the ordinary lixiviation-process, the Russell process, and both processes together. (mixed months). The total expenses, including assaying, drying, crushing, roasting, lixiviating, roasting and refining of the sulphides, and production of bullion 980 fine, while using the Russell process, were $12.08.

It should be observed that the expenses at Cusi are probably double what they would be in a well-constructed mill in the United States.

The Cusi mill was built so as to utilize, in part, old plant, and to work San Bartolo ore by the ordinary process. It was ill-adapted, particularly in roasting and dust-saving facilities, for working the baser San Antonio ore, and in its lixiviating department was unsuited to the Russell process.

For roasted ore, at the Sierra Grande Mill, Lake Valley, the total mill-expenses for 60 tons per day, were estimated by the general manager, Mr. Hadley, at $4.65 per ton.

At Parral, Mexico, the total expenses for the treatment of tailings from ore which had originally been roasted and lixiviated by the ordinary process, were $2.10 per ton for 40 tons per day. For roasted ore, at the rate of 10 tons per day, the total expenses were $9.15.

At Silver Reef, Utah, the total cost of lixiviation, including assaying and general expense, was $1.65 per ton for raw tailings, at the rate of 40 tons per day, and $3.00 to $4.00 for raw ore, at the same rate per day. The corresponding expenses for 75 tons per day would be about $1.25 for raw tailings, and about $2.85 for raw ore.

In a well-constructed lixiviation-mill, the total expenses for treating 75 tons raw ore per day should not exceed $3.00 per ton, and under favorable circumstances would fall as low as $2.50, particularly if the crushing were done by rolls instead of stamps.

§ 1714. *Cost of Chemicals in Stock.*

The following statement regarding chemicals carried in stock for a lixiviation-plant of 80 tons capacity per day, will show the economy of lixiviation as compared with amalgamation. Condi-

tions are assumed as they may actually exist in practice. The table below explains itself.

TABLE No. 1714.

SHOWING COST OF CHEMICALS CARRIED IN STOCK UNDER ASSUMED CONDITIONS.

CHEMICALS.	Cost per Pound, Including Freight. Cents.	Consumed per Ton of Ore. Pounds.	Consumed for 80 Tons per Day. Pounds.	Stock for 60 Days. Pounds.	Cost of Stock for 60 Days. Dollars.
Copper Sulphate	8.0	4.7	376	22,560	1,804.80
Sodium Hyposulphite	4.0	3.7	296	17,760	710.40
Caustic Soda	5.0	4.0	320	19,200	960.00
Sulphur	3.25	2.7	216	12,960	421.20
Sulphuric Acid	3.5	1.0	80	4,800	168.00
Total Tons				38.64	$4,064.40
To this should be added Sodium Hyposulphite in Stock-Solution					150.00
Total					$4,214.40

For amalgamation, a mill of 80 tons daily capacity would have to carry a quantity of quicksilver, costing at least ten times the amount calculated above.

CHAPTER XVIII.

THE RUSSELL PROCESS COMPARED WITH ORDINARY LIXIVIATION, AND WITH AMALGAMATION.*

A. THE RUSSELL PROCESS COMPARED WITH ORDINARY LIXIVIATION.

a. Roasted Ores.

§ 1801. *Results at the Ontario Mill, Park City, Utah, in* 1883–1884.

TABLE No. 1801.

RECORD OF EXPERIMENTS WITH ONTARIO ORE, 1883-84. (ACID WASH-WATER.)

Value of Ore. Oz. Silver per Ton.	Per cent. Extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted in Mill.	Per cent. of Silver Made Insoluble in Ordinary Solution by Caustic Soda.	Difference Between Extraction in Mill and Extraction by Extra-Solution in Assay-Office. Per cent.
AVERAGE RESULTS OF THE FIRST FOUR MILL-TESTS WITH THE ORDINARY PROCESS. (NEITHER ACID NOR EXTRA-SOLUTION WAS USED.)					
85.0	89.0	90.5	81.8	6.2	8.7
AVERAGE RESULTS OF THE SECOND FOUR MILL-TESTS WITH THE ORDINARY PROCESS. (NEITHER ACID NOR EXTRA-SOLUTION WAS USED.)					
86.5	84.3	92.5	66.2	14.6	26.3
AVERAGE RESULTS OF THE NEXT TEN MILL-TESTS, IN WHICH ACID WAS USED BUT NO EXTRA-SOLUTION.					
90.4	82.6	90.0	82.5	0.0	7.5
AVERAGE RESULTS OF SUBSEQUENT MILL-TESTS, IN WHICH BOTH ACID AND EXTRA-SOLUTION WERE USED.					
72.0	87.0	92.6	92.5	0.0	0.1

Table No. 1801 contains the results of Mr. Russell's first lixiviation-experiments at the Ontario Mill, with ore-charges of two tons. For precipitation of the silver, sodium sulphide was used

* The statistics in this chapter are taken from Mr. Daggett's paper, with exception of §1810.

which, on account of defective preparation, contained free caustic-soda. In consequence, the lixiviation-solution assumed a caustic reaction, and the extraction of the silver in the mill was materially lessened. This is shown in the first set of figures in Table No. 1801, the assay-office extraction with ordinary solution being 89.0 per cent., while the mill-extraction was only 81.8 per cent. Samples of the mill-tailings were then treated in the assay-office with ordinary solution, which brought up the extraction of the silver to only 82.8 per cent., instead of 89.0 per cent., showing that lixiviation in the mill with a caustic ordinary solution had rendered insoluble 6.2 per cent. of the silver in the ore. During the next 4 tests, the caustic soda in the stock-solution had increased to such an extent as to cause the mill-extraction to fall to 66.2 per cent., or 18.1 per cent. below the assay-office extraction with ordinary solution. From these mill-tailings, ordinary solution extracted in the assay-office 3.5 per cent. more silver, leaving a discrepancy of 14.6 cent., the amount of silver rendered insoluble by lixiviation in the mill.

Now, the stock-solution was neutralized by addition of sulphuric acid, and, in all subsequent experiments, it was kept neutral, consuming from $\frac{3}{4}$ to 1 pound of sulphuric acid per ton of ore. The next ten tests show the beneficial effect of the acid, the extraction of silver with ordinary solution in the mill differing from that in the assay-office only 0.1 per cent.

Finally, extra-solution was used in addition to sulphuric acid, with results shown at the end of the table. I leave to the reader comparing the last results with previous ones.

§ 1802. *Results at the Sierra Grande Mill, Lake Valley, New Mexico.*

TABLE No. 1802.

RECORD OF MILL-RESULTS, SIERRA GRANDE ORE. (ALKALINE WASH-WATER.)

Per cent. Extraction by Ordinary Solution in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Ordinary Process in Mill.	Per cent. Extraction by Russell Process in Mill if Extra-Solution was Used after the Ordinary.	Per cent. Extraction by Russell Process in Mill Using a Weak Extra-Solution before the Ordinary.
71	81.5	53	74.4	83

Table No. 1802 gives the average mill-results by ordinary lixiviation, and by two methods of the Russell process. The table illustrates: 1st. The difference in mill-results between ordinary lixiviation and the Russell process. 2d. In treatment by the Russell process, the difference between using a warm standard extra-solution, with circulation, after ordinary lixiviation, as if the ore were an acid ore; and a cold extra-solution, low in copper, applied without circulation, and at once, that is, without first using ordinary solution.

As shown in the table, the results of the mill-tests were 30 per cent. of the value of the ore in favor of the Russell process, as compared with ordinary lixiviation; although the difference in extraction between ordinary and extra-solution in the assay-office was only 10.5 per cent. The mill-workings by ordinary lixiviation were below the assay-office results with ordinary solution on account of the injurious effect of the caustic lime in the roasted ore. No separate clean-up was made for the mill-tests by the ordinary process. The clean-up from the Russell process averaged 0.5 to 1.5 per cent. short of that called for by the "apparent" extraction. Although the mill was built for a capacity of 80 tons per day, and the capacity of the lixiviation-plant was shown to be 80 to 100 tons, the crushing-plant failed to reduce more than 20 tons, thus limiting to that figure the amount which could be treated. The average value of the ore for the first two months was about 17 ounces silver per ton, and for the last five months about twelve ounces.

§ 1803. *Results at the Sombrerete Mill, Zacatecas, Mexico.*

Table No. 1803 gives the results of preliminary mill-tests made at Sombrerete to determine the applicability of the Russell process, the ore being roasted in reverberatory furnaces and lixiviated in charges varying from 325 pounds to 4.5 tons. The first three tests were made on very fine and pure pyrites. Sintering of the charge took place during roasting on account of the entire absence of gangue, and the fineness of the pyrites. In consequence, the mill-extraction with extra-solution fell short of the extraction in the assay-office, 5.9 per cent.

TABLE No. 1803.

RECORD OF EXPERIMENTS WITH SOMBRERETE ORE. (ACID WASH-WATER).

Description of Charges.	Weight of Charge.	Value in Ounces Silver Per Ton.	Per cent. Extraction by Ordinary Solution in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Russell Process in Mill.
Finest Pyrites from the Jigs, crushed through an 8-screen—3 Charges	325 lbs.	75.8	73.5	90.9	85.0
Coarse and Fine Jig-Products mixed half and half, crushed through an 8-screen—1 Charge	325 lbs.	65.4	67.4	91.3	90.8
Coarse Jig-Products, crushed through an 8-screen—5 Charges.	325 lbs.	33.4	76.7	86.2	86.5
Normal Mixture of Jig Products, crushed through an 8-screen—4 Charges	325 lbs.	44.9	80.6	90.6	91.3
Normal Mixture of Jig-Products, wet down while red-hot—3 Charges	4½ tons.	35.4	77.5	84.6	80.6
Normal Mixture of Jig-Products, *wet down cold*—5 Charges	4½ tons.	36.5	74.0	89.3	88.0

The fourth set of mill-tests illustrates the injurious effect of wetting down the roasted ore while red-hot, the results in the mill, with extra-solution, falling 4 per cent. below those in the assay-office.

At the time of making the above tests, the results in the mill with ordinary solution were averaging very much below those in the assay-office, so that the actual difference in favor of the Russell process, as compared with the ordinary process, averaged 22.6 per cent. of the value of the ore. The difference in actual extraction in silver was greater, as the clean-up from the ordinary process fell a little short, while that from the Russell process was slightly in excess of the apparent extraction. Since Mr. Watson has had charge of the Sombrerete Mill, both the mill and the assay-office results have been very much higher.

§ 1804. *Results at the Cusihuiriachic Mill, Chihuahua, Mexico.*

Table No. 1804. a. gives the mill-results at Cusi by the Russell process and the ordinary process, during a period of 9 months. The "mixed" months, referred to in the table, are those in which

the extra-solution was used on a portion (about one-half) of the charges. During the other months, either the Russell process or the ordinary process alone was used. The comparison between the two methods is in favor of the Russell process, as follows:

TABLE No. 1804. a.

THE RUSSELL PROCESS COMPARED WITH ORDINARY LIXIVIATION AT THE CUSI MILL.

Months and Process Used.	"Ordinary," "Mixed" or "Russell" Process Months.	Value of Roasted Ore in Ounces Silver Per Ton.	Average Total Time of Lixiviation. Hours.	Per cent. Apparent Extraction in Mill.	Value of Product in Ounces Per Ton.	Total Mill-Expenses Per Ton of Ore.	Total Mill-Product Per Month.	Net Mill-Profit Per Ton.	Net Mill-Profit Per Month.
Jan. 1887.... Ordinary ... Feb. 1887.... Ordinary....	"Ordinary" Months.	35.1	66	78.3	8.681	$13.37	$19,900	$ 7.81	$ 7,336
Sept. 1886.... ⅓ R. P..... Oct. 1886..... ⅔ R. P..... Mar. 1887.... ⅔ R. P..... May, 1887.... ½ R. P.	"Mixed" (or part Ordinary and part Russell Process) Months.	36.8	46	82.3	7.985	$12.62	$27,690	$12.42	$13,724
Nov. 1886.... R. Process.. Dec. 1886.... R. Process.. Apr. 1887.... R. Process..	"Russell Process" Months.	39.9	41	84.8	8.201	$12.08	$42,233	$15.11	$23,436

By the use of extra-solution the time of lixiviation is reduced 34.8 per cent.; the apparent extraction is increased 6.5 per cent.; the expenses per ton are reduced $1.29; the net mill-profit per ton is nearly doubled; the gross mill-product per month is more than doubled, and the net mill-profit per month is more than trebled. It will be noticed that the ore during the months when extra-solution was used, was 4 to 5 ounces higher in value than while the ordinary process was employed. Making corrections accordingly, *i. e.*, reducing the value of the ore to 35.1 ounces, the comparison would be in favor of the Russell process by 50 per cent. greater net profit per ton; 86.2 per cent. additional gross product per month, and 149 per cent. additional net profit per month.

TABLE No. 1804. b.

SIX SPECIAL MILL-RUNS AT CUSI TO DETERMINE THE AMOUNT OF DISCREPANCY BETWEEN "APPARENT" AND "ACTUAL" EXTRACTION IN THE MILL.

No. of Mill-Run.	Ore Treated.	Mesh of Screen on Battery.	Per cent. of Salt Used.	Total Weight of Ore Lixiviated. Tons.	Value of Ore in Ounces per Ton.	Per cent. Extraction by Ordinary Solution in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Apparent Extraction in Mill. Per Cent.	Actual Extraction in Mill. Per Cent.	Condition of Sulphides.
1	San Bartolo and San Antonio	16	9	19.6	39.8	85.5	92.0	85.3	84.2	Roasted.
2	San Bartolo and San Antonio	16	10	18.7	38.4	83.4	91.9	79.7	79.2	Furnace-Dried.
3	San Bartolo and San Antonio	12	11	46.8	83.9	84.8	91.6	82.6	87.9	Furnace-Dried.
4	San Antonio and San Bartolo	12	8	94.	41.7	82.7	90.4	85.9	86.5	Steam-Dried.
5	San Bartolo and San Antonio	26	10	50.	46.9	85.0	89.2	88.5	91.5	Steam-and Furnace-Dried.
6	San Miguel	26	8	50.	53.3	89.0	91.9	90.4	91.8	Steam- and Furnace-Dried.
	Average.					84.8	90.8	86.2	87.9	

NOTE.—The Second Mill-Run was by the Ordinary Process.

TABLE No. 1804. c.

SPECIAL MILL-RUNS BY RUSSELL PROCESS WITH CHARGES ALREADY TREATED BY ORDINARY LIXIVIATION, ON WHICH THE VALUE OF THE TAILINGS COULD NOT BE FURTHER REDUCED BY THE LATTER PROCESS.

Number of Charges Treated.	Number of Tons Treated.	Value of Ore in Ounces Per Ton.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Mill Tailings after Ordin'y Process and before Russell Process.	Mill Tailings after Russell Process.	Mill Per cent. after Ordin'y Process and before Russell Process.	Mill Per cent. after Russell Process.	Difference bet. Ordinary and Russell Process in Mill in oz. Per Ton.	Difference in Per cent. of Value of Ore.	Amt. of Silver called for by Difference in Tailings. Total oz.	Silver Obtained in Furnace-Dried Sulphides. Ounces.	Silver Obtained in Roasted Sulphides. Ounces.	Per cent. Discrepancy based on Dried Sulphides.	Per cent. Discrepancy based on Roasted Sulphides.
51	450	44.0	11.2	81.2	....	13.4	6.5	69.6	85.2	6.9	15.6	No. separate clean-up in sulp's.				
11	97	40.5	10.0	78.1	89.6	15.0	7.2	63.0	83.0	7.8	19.2	"	"	"	"	"
11	82	39.3	12.6	82.5	88.5	15.6	7.8	61.1	81.0	7.8	19.9	640.	747.8	732.	+3.3	+2.9
19	174	48.3	13.5	72.7	86.8	24.5	7.7	49.3	84.1	16.8	34.8	2922.	2890.0	2860.	—1.1	—2.1
92	803	43.0	11.8	78.6	88.3	17.1	7.3	60.7	83.3	9.8	22.4	3562.	3637.8	3592.	+2.2	+0.8

During the nine months referred to, a discrepancy nearly always existed between the apparent and the actual extraction, both while using the ordinary and the Russell process. Part of this (3.2 per cent.) was due to volatilization of silver, and loss by dust in roasting the sulphides. This was remedied by substituting steam-drying for roasting. Another portion (2.8 per cent.) was found to be due to incorrect assaying of the sulphides; another to incorrect determination of the weight of ore lixiviated. Except in the special tests given in Table No. 1804. b. the weight of ore actually lixiviated could not be accurately determined, as there were, at the time, no scales for weighing, and practical difficulties existed in the way of introducing them for general use. The only means of approximately determining the weight of ore was by the cubic foot system. Careful experiments showed that this method gave 6 per cent. more ore than was actually present in the charge. But this approximate method was better than none. At intervals careful experiments were made to determine whether a discrepancy really existed between the apparent (*i. e.*, the extraction calculated from the value of the tailings as compared with the value of the ore) and the actual extraction (*i. e.*, the product in silver). In all, six of these tests were made during a period of nine months, the weight of ore, varying from 18 to 94 tons, being carefully determined. All these tests are given in Table No. 1804. b. The total amount of ore treated in these special tests was 279.1 tons. The average "actual" extraction was 87.9 per cent., which is 1.7 per cent. above the "apparent" extraction. No. 2 (of the 6 special tests) shows both an apparent and an actual extraction of over 79 per cent. by the ordinary process.

On San Antonio ore, or on a mixture of San Antonio and San Bartolo, the extraction by the ordinary process averaged much less than this, the difference between the ordinary and the Russell process being 15 to 34 per cent. of the value of the ore, as shown in Table No. 1804. c.

This set of special mill-runs was made to determine the exact difference caused by the use of the Russell process, as compared with the ordinary process, both in regard to the value of tailings and in actual product in silver. The ore was mixed—San Antonio and San Bartolo. In order to get the best possible results by the ordinary process, the percentage of salt used in chloridizing-roast-

ing was increased, and each charge of ore was lixiviated continuously (some of them for five or six days) by the ordinary process, until for two consecutive days the tailings remained the same, and it was evident that no further reduction in the value of the tailings could be effected. The extra-solution was then turned on the same charges, reducing the value of the tailings 9.8 ounces silver per ton. The increase in the percentage extracted, due to the use of the Russell process, was 22.4 per cent. of the value of the ore. The increase in "actual" extraction caused by extra-solution exceeded the amount called for by difference in value of tailings 2.2 per cent. if the sulphides were dried, and 0.8 per cent., if roasted.

§ 1805. *Results at the Yedras Mill, Sinaloa, Mexico.*

The Russell process has been successfully introduced in the Yedras Lixiviation-Mill, Sinaloa, Mexico. The Yedras ore contains considerable calcite and arsenical minerals. A reliable analysis not being extant, I cannot state the quantities of lime and arsenic. The reduction of the Yedras ore is interesting in many respects. Chloridizing-roasting in Brueckner cylinders, originally provided in the mill, was a failure, and these furnaces had to be replaced by reverberatories. For details on this subject, I refer to Mr. Geo. Rockwell's paper, already quoted in § 10. After crushing by stamps through a No. 20 wire-screen, the ore is roasted in reverberatory furnaces with 7 per cent. salt. On account of the arsenic, the loss of silver by volatilization is considerable, and may rise to 18 per cent., and even higher, if the roasting is conducted so as to obtain a mill-extraction of 90 to 92 per cent. by the Russell process, calculated on the value of the roasted ore. Hence, it is judicious to aim at a lower extraction, keeping the loss of silver in roasting below 7 per cent.

The volume of the first wash-water averages 32; of ordinary solution 100; and of extra-solution 14.5 cubic feet per ton of ore. This quantity of extra-solution is unusually large, the Yedras ore requiring more liquid to saturate a charge than most other ores. Another curious fact, already noted in § 1407, is, that the best results with extra-solution are obtained by allowing it to stand for 12 hours in the ore, without circulation.

The consumption of copper sulphate for extra-solution is 9.6 pounds per ton of ore; the concentration of the stock-solution in sodium hyposulphite is kept at 1.8 per cent.; the rate of lixiviation is 7 inches.*

A good opportunity was offered at Yedras to test the merits of the Russell process, as compared with ordinary lixiviation. A new lixiviation-plant was erected in the mill, the old one being badly constructed and causing loss by leakage. The construction of the new plant, which was especially designed for the Russell process, progressed, however, so slowly that for two months—November and December, 1887—one-half of the ore was still treated in the old plant by the ordinary process, and the other half in the new plant by the Russell process.

Table No. 1805. a. shows the superiority of the Russell process.

TABLE No. 1805. a.

THE RUSSELL PROCESS AND ORDINARY LIXIVIATION AT YEDRAS.

Month.	By which Process Treated.	Value of Roasted Ore Per Ton.	Per cent. by Ordinary Solution in Ass'y Office.	Per cent. by Extra-Solution in Ass. Office.	Per cent. Extraction in Mill.	Additional Extraction per Ton by Russell process.
November.	Ordinary.	56.9 oz.	69.4	82.0	63.7	9.85 oz.
	Russell.	55.7 "	70.4	82.6	81.2	
December.	Ordinary.	61.15 "	71.9	84.0	67.3	8.89 "
	Russell.	58.15 "	67.9	83.8	82.2	

Table No. 1805. b. gives the mill-extraction at Yedras for 7 months.

* According to a letter from Mr. R. D. Rhodes, dated May 6, 1888, the concentration of the stock-solution in sodium hyposulphite is 2.3 per cent. (never more than 2.5 per cent., or less than 2.0 per cent.) The consumption of copper sulphate is 6.5 pounds per ton of ore; the rate of lixiviation from 2 to 4 inches (without vacuum below the filter).

TABLE NO. 1805. b.

MILL-EXTRACTION AT YEDRAS.

Months (1887).	By Ordinary Lixiviation.	By Russell Process.
June	71.1 per cent.	
July	66. "	Process
August	66. "	not in use till
September	73.1 "	November.
October	63.1 "	
November	63.7 "	81.2 per cent.
December	67.3 "	82.2 "

The following statistics have been furnished by Mr. Frank M. Johnson, Superintendent of the Yedras Lixiviation-Mill.

STATISTICS OF ORDINARY LIXIVIATION.

From June 1st, 1887, to February 1st, 1888.

Number of tons treated	5785
Value per ton in silver, ounces	60.67
Per cent. of silver extracted in assay-office with ordinary solution	72.09
Per cent. of silver extracted in the mill	67.12
Total time of lixiviation, hours	92
Chemicals consumed per ton of ore:	
Caustic lime, pounds	9.7
Sulphur, "	4.7
Cost per ton, Mexican coin	$0.66

STATISTICS OF THE RUSSELL PROCESS.

From November 1st, 1887, to February 1st, 1888.

Number of tons treated	1772
Value per ton in silver, ounces	55.3
Per cent. of silver extracted with ordinary solution in assay-office	69.94
Per cent. of silver extracted with extra-solution in assay-office	83.62
Per cent. extracted by the Russell process in the mill	82.44
Total time of lixiviation, hours	76
Chemicals consumed per ton of ore:	
Sodium hyposulphite, pounds	1.4
Copper sulphate, "	9.6
Caustic soda, "	5.5
Sulphur, "	3.6
Cost per ton, Mexican coin	$2.78

It will be seen that ordinary solution extracts in the assay-office 4.97 per cent. more silver than in the mill, and that the mill-results

are 15.32 per cent. in favor of the Russell process. The gain in extraction on ore assaying 60 ounces silver per ton, would be 9.19 ounces or $11.17 Mexican coin, less increased expense of chemicals, $2.12, leaving a net profit of $9.05 Mexican coin.

The most remarkable feature of these statistics is the low consumption of sodium hyposulphite,—1.4 pounds per ton of ore—considering that 9.6 pounds copper sulphate are used per ton of ore for the preparation of extra-solution. In Mr. Daggett's paper it is even stated that the consumption of sodium hyposulphite had been reduced at Yedras from 3.6 pounds to about one-half pound per ton of ore. The explanation given in Mr. Daggett's paper does not meet the case at all.

To arrive at a full understanding of the problem, I offer the following considerations. If the concentration of the stock-solution was 1.8 per cent. in sodium hyposulphite, 14.5 cubic feet of solution would contain 16.3 pounds of this salt. For preparing standard extra-solution, 9.6 pounds copper sulphate would require the presence of 21.6 pounds sodium hyposulphite; hence, 5.3 pounds of this salt would have to be added to the 14.5 cubic feet stock-solution, while the total consumption is only given at 1.4 pounds. If, however, the concentration of the stock-solution in sodium hyposulphite was 2.23 per cent., 1.4 pounds of this salt would suffice to make a standard extra-solution; hence, the statement in Mr. Daggett's paper, that the concentration of the stock-solution was 1.8 per cent. may not be correct.

More difficult of explanation is the fact that with a consumption of 9.6 pounds copper sulphate per ton of ore, only 1.4 pounds sodium hyposulphite were lost. In the preparation of extra-solution, 9.6 pounds sodium hyposulphite are converted into tetrathionate. Assuming that sodium sulphide of low precipitating-coefficient was used, not more than 4.94 pounds sodium hyposulphite per ton of ore would be added to the stock-solution after precipitation of the sulphides.* This quantity added to 1.4 pounds, gives 6.34 pounds, still leaving a difference of 3.26 pounds sodium hyposulphite unaccounted for, not taking into consideration the unavoidable loss of this salt by atmospheric decomposition. Even

* Sodium sulphide prepared from 5.5 pounds caustic soda would contain 4.94 pounds sodium hyposulphite.

an old and highly oxidized sodium sulphide solution would not make up this difference.

My first impression was, that, by inadvertence, a mistake had been made in rendering these statistics. Upon further inquiry, however, I find that this is not the case, as shown by the following official information from Mr. Covington Johnson, late General Manager of the Anglo-Mexican M. Co. The consumption of chemicals at the Yedras Mill, for the months of February and March, 1888, during which the Russell process alone had been in use, was as follows, viz:

	February.	March.
Tons of ore worked	1012	1295
Pounds of chemicals consumed per ton of ore:		
Sodium hyposulphite	1.00	0.74
Copper sulphate	7.47	7.70
Caustic soda	4.39	5.86
Sulphur	2.92	3.39

This unquestionably confirms the fact that the consumption of sodium hyposulphite at the Yedras Mill is unusually low, much lower than in all other mills where the Russell process has been applied, and in contradiction to the arguments I have advanced in my criticism. In referring to Table No. 1712, it will be seen that the consumption of sodium hyposulphite is in all cases—with exception of the Yedras ore—more or less in proportion to the amount of copper sulphate used in the preparation of extra-solution, and in accordance with well known chemical reactions. How, then, can this paradox be explained?

Without chemical research and careful observation of the whole process "in loco," it is, of course, impossible to advance an explanation. I can only draw attention to the fact that the Yedras ore is the only example, so far observed, representing the class of "arsenical alkaline" ores. Although a solution of this problem would not benefit the Yedras Mill financially, it would be highly interesting to the metallurgical chemist.

b. Raw ores.

§ 1806. *Results at the Flagler Reduction-Works, Silver City, New Mexico.*

TABLE No. 1806.

RECORD OF RESULTS, LIXIVIATING BREMEN TAILINGS.

Fineness of the Tailings — Per cent. Passing a Screen of 22,500 Holes per sq. inch.	Average Rate of Lixiviation with vacuum of 14″ of Mercury.	Value in Ounces Silver per Ton.	Per cent. Extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted by Ordinary Solution in Mill.	Per cent. Extracted by Ordinary and Extra-Solution in Mill.
87.8	0.667	11.6	42.5	64.7	38	61.7

Table No. 1806 gives the average results on the raw Bremen tailings, at Silver City, for the first month's run. The average difference between the results of the ordinary process and the Russell process in the mill, was in favor of the latter by 23.7 per cent. of the value of the material treated. During the second month, the average difference was 20.9 per cent., and for the third month, 25.9 per cent. Afterwards, the tailings became too fine for further treatment, the average rate of lixiviation for the second month being only one-seventh of an inch, and for the third month, one-tenth of an inch.

B. The Russell Process Compared with Amalgamation.

a. Roasted ores.

§ 1807. *Results at the Ontario Mill in* 1883–1884.

Table No. 1807 illustrates the difference in mill-results between the Russell process and amalgamation on Ontario roasted ore. The amalgamation-tests were necessarily confined to ore crushed through a 30-mesh screen, and roasted with 16 to 18 per cent. of salt. The lixiviation-tests were made on ore crushed through screens as coarse as 16-mesh, and with all percentages of salt, from 18 down to 0. The table shows that the results of lixiviating ore roasted without salt were only 4 per cent. less than those

by amalgamation of ore roasted with 18 per cent. of salt. Also, that on ore crushed through a 20-mesh screen, and roasted with 12.5 per cent. of salt, the extraction by lixiviation was 4.5 per cent. greater than by amalgamation with finer crushing using 3.5

TABLE No. 1807.*

RECORD OF RESULTS, LIXIVIATING AND AMALGAMATING ONTARIO ORE, 1883–84, (ACID WASH-WATER).

Process Used.	Per cent. of Salt.	Size of Screen.	Per cent. Extracted.
Russell Process	0.	30	84.9
Amalgamation.....	18.2	30	89.4
Russell Process	0.	30	85.7
Amalgamation.....	18.2	30	89.3
Russell Process	9.0	30	96.2
Amalgamation.....	15.9	30	92.5
Russell Process	12.5	20	97.0
Amalgamation.....	15.9	30	92.5
Russell Process	8.0	16	92.0
Amalgamation.....	17.9	30	96.1
Russell Process	12.0	16	97.1
Amalgamation.....	17.3	30	92.5
Russell Process	16.0	16	95.7
Amalgamation.....	17.5	30	95.5
Russell Process	18.0	16	95.1
Amalgamation.....	17.5	30	93.3

per cent. more salt. The average results by lixiviation on ore crushed through a 16-mesh screen, roasted with 12 per cent. of salt, were also higher than those by amalgamation on ore crushed through a 30-mesh screen, roasted with 17.5 per cent. of salt. The amount of water required for amalgamation was 2.3 times as great as that used in lixiviation.

* The weight of ore-charges in these experiments was 2 tons for lixiviation, and 2800 pounds for amalgamation.

§ 1808. *Results at the Marsac Mill, Park City, Utah, in* 1887.

TABLE No. 1808.

RECORD OF RESULTS, LIXIVIATING AND AMALGAMATING DALY ORE, 1887.

No. of Mill-Run.	Total Time covered by Mill-Runs.	Value of Roasted Ore. Oz. Silver per Ton.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
1	October 20th to November 20th, 1887.	33.22	10	93.4	94.4	90.0	88.9
2		35.43	10	87.2	91.0	90.7	88.5
3		43.03	10	89.9	91.2	89.9	87.5
4		40.04	10	88.8	90.8	89.0	86.6
5		38.87	10	89.4	91.6	90.7	88.2
6		39.34	10	86.9	88.9	90.6	86.3
7		40.87	10	88.9	90.9	90.5	90.8
8		42.32	10	87.1	91.0	93.5	90.8
9		35.94	9	88.5	90.7	92.3	90.9
10		40.13	10	88.0	90.5	89.8	91.8
11		39.67	12	89.4	91.6	89.9	91.0
Averages.				88.9	91.1	90.6	89.2

NOTE.—The ore was crushed by stamps through a No. 20 screen.

Table No. 1808 gives the mill-tests made by Mr. W. A. Wilson, Superintendent of the Marsac Mill. In all cases the ore was roasted for amalgamation—not for lixiviation. If the ore had been roasted with reference to the Russell process, the results by the latter would probably have been 2 to 5 per cent. higher. The weight of the charges in lixiviation was 2 tons, and for amalgamation, 1.5 tons. The amount of water used in amalgamation was 7 to 8 times greater than that required for lixiviation. An exact comparison in expenses has not been made; the Marsac mill-expenses are about $11, and the difference will probably be $4 to $6 in favor of the Russell process, the ore being alkaline, and requiring only a small amount of chemicals.

§ 1809. *Results at the Ontario Mill in* 1887–1888.

These mill-tests were also conducted by Mr. Wilson to determine the applicability of the Russell process to Ontario ore. In all the tests, the crushing, the percentage of salt used, and the

roasting were regulated with reference to obtaining the best possible results by amalgamation, and not by lixiviation. Nevertheless, the mill-extraction by the Russell process averages 9.4 per cent. of the value of the ore above the results of amalgamation. This is due to a change in character of Ontario ore, as compared with ore extracted in 1883–1884. Ontario ore has always re-

TABLE No. 1809.

RECORD OF RESULTS, LIXIVIATING AND AMALGAMATING ONTARIO ORE. 1887.

No. of Mill-Run.	Total Time Covered by Mill-Runs.	Value of Roasted Ore. Oz. Silver per Ton.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
1	November 20, 1887, to January 1, 1888.	55.78	11	91.6	91.8	73.6	91.8
2		40.28	11	83.8	91.9	71.3	92.1
3		36.49	13	91.9	94.1	75.5	86.3
4		43.72	14	93.1	94.2	86.3	89.5
5		37.06	13	93.8	94.4	89.6	94.9
6		39.08	12	87.1	91.8	81.1	90.8
7		43.50	12	89.5	92.2	89.6	91.3
8		51.40	14	83.9	89.9	86.8	89.2
9		43.10	15	79.0	91.4	76.6	90.6
10		38.10	16	94.0	95.9	81.8	95.1
11		52.90	15	93.8	94.2	91.3	94.8
Averages.....		43.76		89.2	92.9	82.1	91.5

NOTE.—The ore was crushed by stamps through a No. 26 screen.

quired about three times as much water for leaching as any other roasted ore, yet, the amount of water used in amalgamation is 2.7 times that used in leaching. The Ontario mill expenses are about $13 per ton. Those for lixiviation would be about $5 to $6 less, or a difference in favor of the Russell process of $5 to $6 in expenses, which, together with the additional extraction of 4.11 ounces silver, would make a total net difference of $9 to $10 per ton, in favor of lixiviation.

The following statement gives the average results for January, 1888, by the Russell process and by amalgamation:

Extracted by:	Per cent.
Ordinary solution in assay-office	92.1
Extra-solution in assay-office	93.8
Russell process in mill	93.0
Amalgamation in mill	83.1

Another statement below shows the difference between the extraction with ordinary solution in the assay-office, and amalgamation in the mill, for the last ten days in January.

Ordinary solution in assay-office	92.0
Amalgamation in mill	78.1

The best results by the Russell process on Ontario ore were obtained while using a stock-solution of about 1.5 per cent. concentration in sodium hyposulphite, at a temperature of 90° to 120° F., and with an extra-solution of about the same temperature, using 6½ pounds of copper sulphate per ton of ore. The depth of the dry charge before leaching was 24 inches, and after leaching, 18 inches. For Daly ore, the depth was 22 inches before and 16 inches after leaching. The temperatures of the solutions were the same as for Ontario ore; but the amount of copper sulphate was only 4½ pounds per ton. Although both Ontario and Daly ore yield at present an alkaline first wash-water, the extra-solution can be circulated (and usually is), as if the first wash-water was acid. The amount of gold extracted by the Russell process from both Ontario and Daly ore is the same as by amalgamation. The amount of gold in the ore is, however, quite small, amounting to only 0.1 ounce per ton for shipping-ore, and about one-half that amount for milling-ore.

§ 1810. *Results at the Ontario Mill*, 1888.

April 25th, 1888, Mr. Russell commenced a new series of lixiviation-experiments at the Ontario Mill, of which I can give here only the results of the first set. These experiments were instituted on account of the low extraction of the silver by amalgamation, as already noted in § 1809. Unfortunately, I can not state, from personal observation, in what respect the character of the Ontario ore has changed, as compared with ore mined in 1883–84. From information received, I infer that the ore now contains more or less calcite or limestone in the gangue, and is less rich in sulphurets of the base metals. The lime, being partly converted into caustic lime, imparts to the ore, after chloridizing-roasting, a slightly alkaline reaction, a fact which explains the low extraction of the silver by amalgamation. Whether experiments have been

made to correct this or not, I do not know. The fact remains, that with very high chlorinations of the silver in the roasted ore, the amalgamation is exceedingly low.

In the following experimental mill-run, the ore was crushed in the battery through a No. 26 screen, and roasted in a Stetefeldt furnace with from 9 to 16 per cent. salt. The roasted ore was most carefully divided and weighed, so that lixiviation-vats and pans received material of exactly the same character, richness and quantity, namely, 50 tons each.

The roasted ore contained: *

Silver per ton of ore:	52.4 Ounces.
Salts soluble in water:	12.5 per cent.
Salts soluble in water and sodium hyposulphite:	16.2 "
Silver extractable in assay-office by ordinary solution:	91.6 "
Silver extractable in assay-office by extra-solution:	93.5 "

Amalgamation in pans was conducted in the usual way. For lixiviation, a new vat of 8 tons maximum capacity was added to the existing experimental plant, and also a filter-press.

Amalgamation.

The clean-up resulted as follows:

Silver in bars:	34.53 Oz. p. t., or 65.9	per cent.
Silver left in tailings:†	17.76 " 33.9	"
Loss not accounted for:	0.11 " 0.2	"
	52.40 Oz. 100.00	per cent.

Lixiviation.

The lixiviation was conducted as follows:

Leaching with water being completed, the ore was first treated with ordinary solution, then by extra-solution, followed by ordinary.

The silver from the wash-water was precipitated either by scrap-iron, after addition of sulphuric acid, or by sodium sulphide.

Lead and calcium were precipitated by sodium carbonate.

* The ore contained $1.03 gold per ton. Of this, 54.4 per cent. were actually extracted by lixiviation, and 53.3 per cent. by amalgamation.

† This figure refers to the ounces of silver left for each ton of roasted ore. The tailings as such were, of course, higher in value, namely, in proportion to the concentration caused by leaching out soluble salts.

Silver was precipitated separately from the ordinary and extra-solution by sodium sulphide, so as to obtain exact figures regarding the effect of the extra-solution in extracting silver. The stock-solution—of 1.5 to 2.7 per cent. concentration in sodium hyposulphite—was kept acid by addition of a small amount of sulphuric acid.

It would not interest the reader to enter into all statistics regarding the consumption of chemicals, volumes of solutions, time of lixiviation, etc., which do not offer anything novel, wherefore I proceed at once to give the final results.

RESULTS OF LIXIVIATION.

Silver in products:	42.75 Oz. p. t.,	or 81.6 per cent.
Silver left in tailings:	7.50 "	14.3 "
Silver run to waste in first wash-water:	0.44 "	0.84 "
Loss unaccounted for:	1.71 "	3.26 "
	52.4 Oz.	100.00 per cent.

DISTRIBUTION OF SILVER IN PRODUCTS.

Precipitate from first wash-water:	12.80 Oz. p. t.,	or 24.43 per cent.
Sulphides from ordinary solution:	26.20 "	50.00 "
Sulphides from extra-solution:	2.22 "	4.24 "
Silver in unwashed lead carbonate.	0.74 "	1.41 "
Clean-up of tanks, and silver under filters of vats:	0.79 "	1.52 "
	42.75 Oz.	81.6 per cent.

DIFFERENCE BETWEEN AMALGAMATION AND LIXIVIATION.

The actual clean-up is 8.22 ounces silver per ton, or 15.7 per cent. in favor of lixiviation.

Amalgamation falls 25.7 per cent. short of the assay-office lixiviation-test with ordinary solution.

The Russell process falls short of the assay-office lixiviation-test with extra-solution:

11.9 per cent. in actual extraction.
7.8 " in apparent extraction.

The following points attract our attention:

1st. The loss of silver unaccounted for in lixiviation of 1.71 ounces silver per ton, or 3.26 per cent., may have been caused by incorrect tailings samples, or by more silver running to waste in the first wash-water than stated above, or by other circumstances.

2d. We note a considerable discrepancy between mill-work and assay-office lixiviation-tests, both in amalgamation and lixiviation. Considering the character of the ore, this is not surprising so far as amalgamation is concerned. Regarding lixiviation, it is rather abnormal. I would suggest treatment of the Ontario ore as if it were an alkaline ore—which it really is, if I am correctly informed—by applying a weak extra-solution first, immediately after the first wash-water.

3d. The silver actually extracted by the first wash-water is unusually large. Most likely, the decomposition of the salt in chloridizing-roasting is very incomplete, leaving most of it unchanged. Besides, on account of the small quantity of sulphurets in the ore, calcium chloride may exist in the leaching-brine. Finally, the vats were charged with hot ore, thus considerably increasing the solvent energy of the first wash-water.

Before concluding this paragraph, I must draw attention to another fact.

Every one of my professional brethren knows how silver-assays are made in the West, and that the silver found falls considerably short of the actual contents in the ore. For this reason Mr. Russell did not accept the ordinary mill-assays as a basis for his calculations, but made a careful assay of the sample representing the average of 100 tons treated, and to this he added the silver contained in the slag of the crucible-assay, and that absorbed by the cupel. Here are the figures:

Average of pulp-samples of roasted ore by ordinary mill-assay	46.74	oz. p. t.
Result by careful assay of the same	50.00	" "
Silver in slag and cupel	2.40	" "
Total	52.4	" "

In the same way the assays of the tailings—resulting in the mill and in the assay-office lixiviation-tests—were corrected.

If Mr. Russell had based his calculations upon the mill-assay of 46.74 ounces—and who could have blamed him for doing it—he would have shown to the grateful stockholders of the Ontario Company an actual extraction by lixiviation of 91.46 per cent., and by amalgamation of 73.87 per cent., while the difference in favor of lixiviation would have been 17.59 per cent.

There is food for reflection, oh, Muscular Amalgamator! Where would thy high percentage of extraction be, my muscular friend of the A—— or C—— or G—— M—— mill, if thou wert to base results on truth and not on fiction? And you, my scientific smelting-friends in Colorado, what would become of your "plus silver," and of your small losses in lead?

POSTSCRIPTUM.

On June 25th, 1888, I received the following information from Mr. Russell regarding the discrepancy between mill-extraction by lixiviation and assay-office lixiviation-tests. It appears that Ontario ore, as it is constituted now, requires an unusually large volume of ordinary solution (of 2 per cent. concentration in sodium hyposulphite), before applying the extra-solution, in order to obtain a high extraction of the silver. This will become apparent from the following figures:

Volume of ordinary solution used before extra-solution.	Corresponding extraction by lixiviation.
For Deep Charges.	
55 cubic feet.	72.7 per cent.
68 " "	87.1 " "
76 " "	90.9 " "
For Shallow Charges.	
68 cubic feet.	82.8 per cent.
105 " "	85.6 " "
115 " "	91.4 " "

The second run—which is not yet completed—has given, so far, an apparent extraction of 79 per cent. by amalgamation, and 90 per cent. by lixiviation. The ore for the second run has been roasted with addition of $\frac{3}{4}$ to 1 per cent. sulphur, which accounts for the higher extraction by amalgamation.

On June 30th, 1888, I received the following additional information from Mr. Russell regarding his experiments with Ontario ore. He writes:

"I find that method 10 [see Table No. 1405] is the proper way to lixiviate Ontario ore. Mr. Wilson's former good results, treating it as an acid ore, made me think that it must be an exception to the rule for alkaline ores, but I now find that it is not. The mill-results [92.3 per cent.] come now within 1.7 per cent. of the extraction with extra-solution in the assay-affice. The addition of sulphur to the ore before roasting has raised the amalgamation to 80 per cent.*

"I also find that with this proper treatment the total volume of ordinary solution can be reduced to 100 cubic feet per ton of ore.

"The consumption of the first wash-water for Ontario ore, as constituted now, is only from 40 to 54 cubic feet per ton of ore, provided the charges in the vats are deep."

§ 1811. *Results at the Sierra Grande Mill, Lake Valley, New Mexico.*

TABLE No. 1811.

RECORD OF RESULTS, LIXIVIATING AND AMALGAMATING SIERRA GRANDE ORE.

Per cent. Extraction by Ordinary Solution in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Amalgamation in Mill.	Per cent. Extraction by Ordinary Lixiviation in Mill.	Per cent. Extraction by Russell Process in Mill if Extra-Solution was Used after the Ordinary.	Per cent. Extraction by Russell Process in Mill Using a Weak Extra-Solution before the Ordinary.
71	81.5	70.6	53	74.4	83

Table No. 1811 gives the mill-extraction by the Russell process and by amalgamation, the difference in favor of the Russell process being 12.4 per cent. of the value of the ore. The lixiviation-charges were 18 to 20 tons, while the amalgamation-charges were 1.5 to 2 tons. The amount of water for amalgamation was about eight times that used in leaching. The difference in expense could not be ascertained, as the amalgamation was at the rate of only 3 or 4 tons per day. In this table is illustrated a fact which will be again noted, namely, the percentage of silver

* It is rather astonishing that a remedy of this kind has not been tried long ago to improve the amalgamation at the Ontario Mill. The addition of sulphur is not a new idea. A great deal more effective is pyrites of iron, especially if it contains some pyrites of copper. (The Author.)

obtained by amalgamation in the mill agrees very closely with the percentage obtained by ordinary solution in the assay-office, but bears no relation to the percentage extracted by extra-solution in the assay-office, while the percentage obtained in the mill by extra-solution agrees with the corresponding assay-office test.

§ 1812. *Results at the Cusihuiriachic Mill, Chihuahua, Mexico.*

TABLE No. 1812.

RECORD OF RESULTS, LIXIVIATING AND AMALGAMATING SAN ANTONIO, SAN MIGUEL, AND SAN BARTOLO ORE.

Name of Ore Treated.	Per cent. of Salt.	Value of Ore. Oz. Silver per Ton.	Per cent. of Salts Soluble in Water.	Per cent. by Ordinary Solution in Assay-Office.	Per cent. by Extra-Solution in Assay-Office.	Number of Tons Treated.	Apparent Extraction in Mill. Per cent.	Actual Extraction in Mill. Per cent.	By which Process.
San Antonio and San Bartolo	10.	46.96	16.5	85.0	89.2	50	88.5	91.5	Russell Process.
						50	86.3	79.6	Amalgamation.
San Miguel	8.	53.08	9.4	89.0	91.9	50	90.4	91.8	Russell Process.
						50	87.5	80.8	Amalgamation.

NOTE.—The ore was crushed by stamps through a No. 26 screen.

Table No. 1812 gives the difference in mill-results, both in apparent and actual extraction, between the Russell process and amalgamation. The ore for each test was weighed with the utmost care. In this case also, the crushing, percentage of salt, and the roasting were conducted solely with reference to obtaining the best results by amalgamation. The results show, in the case of amalgamation, a discrepancy between the apparent and the actual extraction, the latter being 6.7 per cent. less than the former. The results of lixiviation, on the other hand, show a *plus* discrepancy, the actual extraction averaging 2.2 per cent. more than the apparent. The actual extraction in silver is in favor of the Russell process by

11.4 per cent. of the value of the ore or 5.7 ounces per ton. The difference in expenses between the Russell process and amalgamation is less for Cusi than for most other ores, being about $2.50 per ton in favor of lixiviation, making a total net difference of over $8 per ton in favor of the Russell process.

b. Raw ores.

§ 1813. *Results at the Big Sandy Mill, Arizona, Comparing the Russell Process with Combined Raw Amalgamation and Concentration.*

TABLE No. 1813.

RECORD OF RESULTS AT THE BIG SANDY MILL.

Number of Run.	Description of Run.	Value Oz. Silver per Ton.	Per cent. Extraction by Concentration.	Per cent. Extraction by Amalgamation.	Total Mill-Extraction.	Per cent. Extraction by Ordinary Solution in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.
I.	4 Days Run without Chemicals..	21.8	28.7	25.3	54.0 pr ct.	49.0	85.1
	3 Days Run with Chemicals.....	21.5	8.7	45.3	54.0 "	50.5	92.5
	Amalgamation by Boss Process *before* Concentration.						
II.	4 Days Run	20.8	43.0		43.0 "	No lixiviation-tests made.	
	Concentration *alone.*						
III.	6 Days Run with Chemicals	30.2	63.7	51.9	53.2 "	50.4	91.5
	Amalgamation by Boss Process *before* Concentration. Coarse Ore Concentrated, Fine Ore Amalgamated.						
IV.	3 Days Run with Chemicals '.....	25.0	44.8	7.6	52.4 "		87.8
	Amalgamation by Boss Process *after* Concentration.						
V.	4 Days Run with Chemicals	20.7	34.4	28.8	63.2 "	No lixiviation-tests made.	
	Amalgamation by *Charyes* after Concentration.						

All the foregoing comparisons have been based on mill-results. On the ore referred to in Table No. 1813, the data for which were furnished by Mr. G. J. Rockwell, no tests have yet been made on a large scale by lixiviation with extra-solution; hence, the comparison is between mill-results by amalgamation and concentra-

tion, on the one hand, and assay-office results by the Russell process on the other. The previous comparisons have been on roasted ore, but this is on raw ore. As no difficulty has been experienced elsewhere in obtaining as good results (within 2 to 3 per cent.) in the mill as in the assay-office on raw ore by lixiviation with extra-solution, there is no reason to expect a material difference in this case. The table shows in Run I. a difference in favor of the Russell process of 53.5 per cent. of the value of the ore, as compared with the Boss process, and of 34.8 per cent. of the value of the ore, as compared with the combined Boss process and concentration. In Run II. lixiviation-tests with extra-solution were not made. In Run III., the difference is in favor of lixiviation as compared with the Boss process, by 39.6 per cent. of the value of the ore, and 38.3 per cent., as compared with the combined Boss process and concentration. Run IV. shows in favor of the Russell process 43 per cent. when compared with concentration, or 35.4 per cent. compared with concentration and Boss process combined.

The average difference in assay-office results with ordinary and extra-solution is in favor of the extra-solution by 38 per cent. of the value of the ore. Here also, as has been noted in the case of roasted ore, the amalgamation-results agree closely with the results by ordinary solution in the assay-office.

§ 1814. *Relation of Mill-Results by Amalgamation and by the Russell Process, to Results respectively obtained in the Assay-Office with Ordinary and Extra-Solution.*

In making the comparison between mill-results by amalgamation and by the Russell process, the relation of the former to the tests with ordinary solution, and of the latter to the tests with extra-solution in the assay-office, has been noted. Table No. 1814 gives the results of mill-runs on five ores of different character by amalgamation and by the Russell process, and the corresponding results in the assay-office with ordinary and extra-solution. The extractions from San Antonio and San Miguel ore are actual clean-ups, the others apparent extractions in the mill. The mill-

results by amalgamation average 80.7 per cent., which is 3.9 per cent. below the test with ordinary solution in the assay-office. On the other hand, the mill-results by the Russell process average 89.4 per cent., or 0.1 per cent. above the extraction with extra-solution

TABLE No. 1814.

RESULTS OF MILL-RUNS COMPARED WITH ASSAY-OFFICE LIXIVIATION-TESTS.

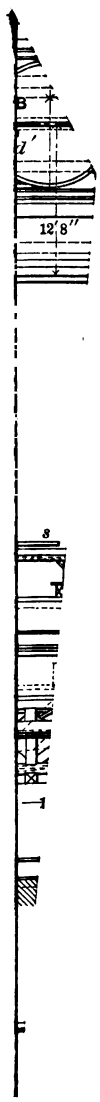
Name of Ore.	Per cent. Extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
Lake Valley	71.0	81.5	70.6	83.0
San Antonio and San Bartolo	85.0	89.2	79.6	91.5
San Miguel	89.0	91.9	80.8	91.8
Daly	88.9	91.1	90.6	89,2
Ontario	89.2	92.9	82.1	91.5
Average	84.6	89.3	80.7	89.4

in the assay-office, and 8.7 per cent. above the results of amalgamation in the mill. These figures indicate that the extraction of silver by amalgamation from roasted ores, is almost as dependent on a good chloridizing-roasting as is the ordinary lixiviation-process. The Russell process, on the contrary, may be entirely independent of a perfect chloridizing-roasting. That, however, a high chlorination of the silver is not always followed by an equally high extraction by amalgamation, becomes evident from the statistics of the Ontario Mill, recorded in § 1809 and § 1810.

POSTSCRIPTUM.

It has been decided to add a lixiviation-plant of 1000 tons monthly capacity to the Marsac Mill for the treatment of Daly ore.

July 6, 1888.


B
12'8"
8
1

INDEX.

A

	Page.
ACID ORES. Definition of	134
" " Lixiviation of	161
AIR-COMPRESSOR for raising solutions	125
" " Cost of	190
ALKALINE ORES. Definition of	134
" " Lixiviation of	163
ARGENTIC HYPOSULPHITE DOUBLE-SALTS	18
ARSENICAL ALKALINE ORES. Definition of	134
" " " Lixiviation of	163
AURIC HYPOSULPHITE DOUBLE-SALTS	18

B

BIG SANDY MILL. Results at the	221
BOILER. Separate — for the lixiviation-plant	129
" The Hazelton	187
BUCKEYE ENGINE	187
BULLIONVILLE MILL. Introduction of the Russell process at the	6

C

CALCIUM CHLORIDE. Solubility of — in water	65
CALCIUM HYPOSULPHITE	14
" " Deterioration of — solution	89
" " Formation of	75
" " Solvent energy of — for silver chloride	89
" " Solvent energy of — for gold	90
CALCIUM SULPHATE. Solubility of — in water	66
" " Solubility of — in brine	67
" " Solubility of — in sodium hyposulphite	32
" " Influence of — on the solubility of silver chloride in sodium hyposulphite	25
" " Precipitation of — by calcium sulphide	87

		Page.
CALCIUM SULPHIDE.	Mode of preparation of	74
" "	Precipitation of — together with sulphides in lixiviation	175
" "	Precipitating-coefficient of — for silver	78
" "	Solution in mill	144
CAUSTIC LIME.	Consumption of — in making calcium sulphide	144
" "	Influence of — on the solubility of silver-compounds in sodium hyposulphite	25
" "	Solubility of — in water	66
" "	Solubility of — in sodium hyposulphite	33
CAUSTIC SODA		15
" "	Analysis of	16
" "	Commercial rating of	15
" "	Consumption of — in making sodium sulphide	142
" "	Cost of	17
" "	Influence of — on the solubility of silver-compounds in sodium hyposulphite	25
CHARGING	lixiviation-vats	145
CHEMICALS.	Consumption of — in lixiviation	194
"	in stock	196
CIRCULATING.	Definition of	134
"	standard extra-solution	141
"	Effect of — extra-solution on acid and arsenical alkaline ores	163
COPPER.	Solubility of metallic — in sodium hyposulphite	20
COPPER SULPHATE.	Consumption of — in preparing extra-solution	141, 195
CORTEZ MILL		27, 29
CUPRIC CARBONATE.	Solubility of — in sodium hyposulphite	28
CUPRIC HYDROXIDE.	Solubility of — in sodium hyposulphite	30, 31
CUPROUS CARBONATE.	Solubility of — in sodium hyposulphite	28
CUPROUS CHLORIDE.	Solubility of — in water	66
" "	Solubility of — in brine	67
" "	Solubility of — in sodium hyposulphite	31
CUPROUS HYDROXIDE.	Solubility of — in sodium hyposulphite	30
CUPROUS HYPOSULPHITE DOUBLE-SALTS		35
CUSIHUIRIACHIC MILL.	Introduction of the Russell process at the	7
" "	Effect of wetting down roasted ore while red-hot at the	10
" "	Cost of precipitating sulphides with sodium and calcium sulphide at the	86
" "	Refining sulphides at the	177
" "	Results at the	201, 220

D

DRYING.	Advantages of perfect — of ore before crushing	184
"	of sulphides	177
DRYING-CHAMBER	for sulphides	129

E

		Page.
ENGINE. Buckeye		187
EXTRA-SOLUTION. Chemistry of the		35
" "	in the mill	139
" "	Decomposition of — by heating	38
" "	Decomposition of — at ordinary temperature	40
" "	Effect of — on silver-compounds	44
" "	Effect of — on raw ores	49
" "	Effect of — on ores oxidized by roasting	54
" "	Effect of — on ores roasted with salt	59
EXTRA-SOLUTION OF STANDARD COMPOSITION		36
" "	Rules for preparing — in the mill	139
" "	Decomposition of — by heating	38
" "	Decomposition of — at ordinary temperature	39

F

FILTER-PRESS		126
" "	Cost of the	190
" "	Manipulation of the	176
FLAGLER REDUCTION-WORKS.	Introduction of the Russell process at the	6
"	" Results at the	210
FUEL. Consumption of — in lixiviation		193

G

GOLD.	Solubility of metallic — in sodium hyposulphite	19
"	Solubility of metallic — in extra-solution	47
"	Extraction of — by lixiviation	63
"	Solvent energy of various hyposulphite salts for	90
GOLD SULPHIDE.	Solubility of—in sodium hyposulphite	21
" "	Solubility of — in extra-solution	48

H

HAZELTON BOILER	187
History of the lixiviation-process	1
" of the Russell process	5

I

INDEX to parts of plant, plates I. and II.	133

K

	Page.
KNOWLES PUMPS	125
" " Cost of	187, 190
KORTING EJECTOR	123
" " Cost of	190
" " Influence of — on the rate of lixiviation	150
KROM'S DRY CONCENTRATION	8
KROM'S ROCK-BREAKERS	184
" " Cost of	187
KROM'S ROLLS. Advantages of crushing by	8
" " Capacity of	183
" " Cost of	187

L

LABOR in a lixiviation-mill	192
LABORATORY WORK	91
LEAD. Solubility of metallic — in sodium hyposulphite	21
" Solubility of metallic — in extra-solution	47
" Separation of — from silver and copper	28, 30
" DISTRIBUTION of — in a lixiviation-solution	165
" Precipitation of — with sodium carbonate	172
" Precipitation of — with caustic lime	31
LEAD ANTIMONATE. Solubility of — in sodium hyposulphite	32
" " Solubility of — in leaching-brine	72
LEAD CARBONATE. Solubility of — in sodium hyposulphite	28
" " Precipitation of — with sodium carbonate	172
LEAD CHLORIDE. Solubility of — in water	66
" " Solubility of — in brine	67
LEAD HYDROXIDE. Solubility of — in sodium hyposulphite	30
" " Precipitation of — by caustic lime	31
LEAD SULPHATE. Solubility of — in sodium hyposulphite	31
" " Solubility of — in leaching-brine	71
" " Influence of — on the solubility of silver chloride in sodium hyposulphite	24
LEAD AND SODIUM HYPOSULPHITE DOUDLE-SALTS	18
LIME, CAUSTIC. Solubility of — in water	66
" " Solubility of — in sodium hyposulphite	33
" " Consumption of — in preparing calcium sulphide	144
" " Influence of — on the solubility of silver-compounds in sodium hyposulphite	25
LIXIVIATION-MILL. Estimated cost of running a	192
LIXIVIATION-PLANT	111
" " Arrangement of the	132
" " Cost of the	189

		Page.
LIXIVIATION-PROCESS.	Ores suitable to treatment by the	8
" "	Fineness of ore crushed for the	9
" "	Testing ores regarding their fitness for the	106
LIXIVIATION-SOLUTIONS.	General rules for handling the	147
" "	Deterioration of the	88
" "	Constitution of the — before precipitation	171
LIXIVIATION-TESTS.	With ordinary solution	93
" "	With extra-solution	95
" "	With extra-solution for raw ores	101
" "	With extra-solution for roasted ores	98
LIXIVIATION-VATS		114
" "	of cement-concrete	118
" "	Cost of	189

M

MAGNESIUM HYPOSULPHITE.	Solvent energy of — for gold	90
MARSAC MILL.	Results at the	212
MECHANICAL STIRRERS		119
MILL-CONSTRUCTION.	Details of	185
MILL-PLANT.	Estimated cost of	187
MOUNT CORY MILL.	Lead-precipitate at the	31

O

ONTARIO MILL.	Experiments lixiviating ores after oxidizing-roasting at the	54
" "	Monthly averages of lixiviation-tests at the	60
" "	Results at the	198, 210, 212, 214
ORDINARY SOLUTION.	See Stock-solution.	

P

PAINTING vats and tanks		191
PARRAL.	Introduction of the Russell process at	7
POTASSIUM HYPOSULPHITE.	Solvent energy of — for gold	90
" "	Solvent energy of — for silver chloride	89
PRECIPITATING-COEFFICIENTS	of sodium and calcium sulphide for silver	78
" "	The same for lead and copper	83
" "	Theory of high — of sodium sulphide	78
" "	Circumstances influencing the value of the — of sodium sulphide	77
PRECIPITATING-TANKS		119
" "	For lead	121
" "	For wash-water	159

Page.

PRESSURE-TANK 129
" " Cost of 190
" " Manipulation of the 176
PUMPS for solution and sluicing tailings 125

R

RATE OF LIXIVIATION 148, 150
RAW ORES. Lixiviation of 160
ROASTED ORES. Lixiviation of acid 161
" " Lixiviation of alkaline 163
" " Lixiviation of arsenial alkaline 163
REFINING SULPHIDES 177
REFINERY FOR SULPHIDES. Cost of 190
ROESSLER CONVERTOR 179
ROCK-BREAKERS 184
ROLLS. Advantage of crushing by 9
" Capacity of Krom's 183
RUSSELL PROCESS. The 3
" " History of the 5
" " Ores suitable to treatment by the 8
" " The — compared with amalgamation 4, 210
" " The — compared with ordinary lixiviation 4, 198

S

SALT. Determination of — in crushed ore 92
SELLERS' HOIST. Cost of 187
SHELF-DRYKILN 188
SIERRA GRANDE MILL. Introduction of the Russell process at the 6
" " " Effect of wetting down roasted ore while red-hot at the 10
" " " Results at the 199, 219
SILVER. Solubility of metallic — in sodium hyposulphite 19, 20
" Solubility of — in extra-solution 46
" Solubility of — compounds in sodium hyposulphite influenced by caustics 25
" Dissolved in first wash-water 156, 157
" Distribution of — in a lixiviation-solution at different periods of the process 165
" Loss of — by volatilization in roasting 108
SILVER ANTIMONATE. Solubility of — in sodium hyposulphite 22
" " Solubility of — in extra-solution 49
" " Solubility of — in brine 71
" " In ores after chloridizing-roasting 57
" " In tailings 23

		Page.
SILVER ARSENATE.	Solubility of — in sodium hyposulphite	22
" "	Solubility of — in extra-solution	49
" "	Solubility of — in brine	71
" "	In ores after oxidizing-roasting	57
" "	In tailings	23
SILVER CARBONATE.	Solubility of — in sodium hyposulphite	28
SILVER CHLORIDE.	Solubility of — in sodium hyposulphite	23
" "	The solubility of — in sodium hyposulphite influenced by the sulphates of lead, calcium and sodium	24, 25
" "	Solubility of — in brine	68
" "	Solubility of — in various chlorides	69
" "	Solubility of — in leaching-brine	71
SILVER OXIDE.	Solubility of — in sodium hyposulphite	21
SILVER REEF.	Introduction of Russell process at	5
SILVER SULPHIDE.	Solubility of — in sodium hyposulphite	21
" "	Solubility of — in extra-solution	47
" "	On top of charge in lixiviation-vats	152
SILVER TETRATHIONATE		43
SODA-ASH		14
" "	Purification of	28
" "	Commercial rating of	15
SODA, CAUSTIC		15
" "	Analysis of	16
" "	Commercial rating of	15
" "	Cost of	17
" "	Consumption of — in making sodium sulphide	142
" "	Influence of — on the solubility of silver-compounds in sodium hyposulphite	25
SODIUM ANTIMONATE.	Solubility of — in water	67
" "	Solubility of — in sodium hyposulphite	34
SODIUM BI-CARBONATE	for neutralizing a caustic stock-solution	27
" "	As precipitant for lead	29
SODIUM CARBONATE		14
" "	Solution in mill	142
" "	Tanks	123
SODIUM CHLORIDE.	Solubility of — in water	65
SODIUM HYPOSULPHITE		13
" "	Cost of	16
" "	Formation of	75
" "	Determination of — with iodine	102
" "	Economy of producing — by oxidation of sodium sulphide	76
" "	Solvent energy of — for gold	90
" "	Solvent energy of — for silver chloride	24
" "	Percentage of — in stock-solution	134
" "	Loss of — in stock-solution	136
" "	Deterioration of — solution by the atmosphere	88

		Page.
SODIUM SULPHATE.	Solubility of — in water	65
" "	Influence of — on the solubility of silver chloride in sodium hyposulphite	25
SODIUM SULPHIDE.	Chemistry of	74
" "	Precipitating-coefficient of — for silver	78
" "	Precipitating-coefficient of — for lead and copper	83
" "	With high precipitating-coefficient	80
" "	Precautions in preparing	81
" "	Tanks	122
" "	Cost of — tanks	190
" "	Effect of diluting — a solution	82, 84
" "	Solution in the mill	142
SODIUM TETRATHIONATE.	Formation of	35
SOLUBLE SALTS.	Determination of — in roasted ore	92
SOLUTION-SUMP		121
SOLVAY SODA		14
" "	Cost of	17
SOMBRERETE MILL.	Introduction of the Russell process at the	7
" "	Results at the	200
STANDARD EXTRA-SOLUTION.	See extra-solution.	
STETEFELDT FURNACE.	Advantages of the	9, 10
" "	Capacity of the	183
" "	Cost of the	187
" "	Oxidizing-roasting in the	54
" "	Loss of silver in roasting in the	108
STOCK-SOLUTION	in the mill	134
" "	Caustic	138
" "	Volume of — per ton of ore	135
STORAGE-TANKS	for stock-solution	112
" "	for sulphides	121
SULPHIDES.	Assaying of — for silver	105
"	Assaying of — for gold	106
"	Precipitation of	174
"	Cost of precipitating	175
"	Drying of the	177
"	Mechanical handling of the	176
"	Refining of the	177
"	Selling — to smelters	181
"	Value of	157, 175
SULPHUR		16
"	Consumption of — in making sodium sulphide	143
"	Consumption of — in making calcium sulphide	144
SULPHURIC ACID		16
" "	For neutralizing a caustic stock-solution	26
" "	For the first wash-water	159
" "	Consumption of — in lixiviation	195

T

		Page.
TAILINGS.	Discharging and sampling	152
"	Extraction of silver from — by ordinary and extra-solution	62
"	Silver arsenate and antimonate in — from amalgamation	23
"	Lixiviation of Bremen	210

V

VOLUME (of solution) to saturate.	Definition of	134
"	Of stock-solution per ton of ore	135
"	Of stock-solution used in mill	138
VOLUMETRIC	determination of sodium hyposulphite with iodine	102

W

WATER.	Consumption of — in lixiviation	195
"	Consumption of — in sluicing tailings	152
WASH-WATER.	Chemistry of the	64
" "	The first	154
" "	Silver dissolved in	156, 157
" "	Distribution of silver in	156
" "	Precipitate from	157
" "	Treatment of the	158
" "	Precipitating-tank	159
" "	The second	151
WEAR AND TEAR	of a lixiviation-mill	194
WETTING DOWN.	Effect of — roasted ore while red-hot	10

Y

YEDRAS MILL.	Introduction of the Russell process at the	7
" "	Results at the	205

INDEX TO DRAWINGS IN TEXT.

	Page.
Fig. 1. Pressure-Tank	128
Figs. 2 and 3. Drying-Chamber for Sulphides	130
Figs. 4 and 5. Ditto	131
Figs. 6 and 7. Precipitating-Tank	120
Figs. 8, 9, 10 and 11. Gate for Sulphides	120
Figs. 12 and 13. Lixiviation-Vat	116
Figs. 14 and 15. Details of Filter in Lixiviation-Vat	120
Fig. 16. Cast-Iron Flange for Lixiviation-Vats, etc.	120
Fig. 17. Cast-Iron Flange for Storage-Tanks	120
Fig. 18. Cast-Iron Flange for Precipitating-Tanks and Solution-Sumps	120
Fig. 19. Cast-Iron Flange for Sodium Sulphide Mixing-Tank and Storage-Tanks	120
Figs. 20 and 21. Flange-Connection for Lead-Pipe and Iron-Pipe	112
Figs. 22 and 23. Sodium Sulphide Mixing-Tank	122
Figs. 24 and 25. Sodium Sulphide Storage-Tank	122
Fig. 26. Korting Ejector	124
Fig. 27. Filter-Press	126
Fig. 28. Tank for Precipitating Wash-Water	159
Fig. 29. Ditto	159
Fig. 30. Roessler Convertor	180

SUPPLEMENT No. 1,

TO

"The Lixiviation of Silver-Ores with Hyposulphite Solutions."

---

§ 102. *Calcium Hyposulphite.*

THE statement that a solution of calcium hyposulphite, if heated above 60° C., decomposes to gypsum and sulphur is not correct. It is quoted from Graham-Otto's Lehrbuch der Chemie, vol. ii, page 531, edition of 1865, and must be a typographical error. In the decomposition of calcium hyposulphite, the sulphite, $CaSO_3 + 2aq.$, not the sulphate, $CaSO_4 + 2aq.$, is formed.

$$CaS_2O_3 = CaSO_3 + S.$$

If, however, the solution is slowly heated under access of air, sulphate may result besides the sulphite.

I am not the only one who is guilty of having copied this error; it is found, for instance, in Gruetzner's Augustin'sche Silberextraction, page 90. The same correction should be made in § 602.

§ 219. *Neutralizing a Caustic Stock-Solution with Sodium Bi-Carbonate.*

If to a neutral sodium hyposulphite solution, containing calcium-salts, a concentrated solution of sodium bi-carbonate is added, a precipitate of calcium bi-carbonate, $CaH_2(CO_3)_2$, is formed, which readily dissolves again upon stirring. A slight turbidity, however, remains. This is due to the presence of one or two per cent. of Na_2CO_3 in the commercial sodium bi-carbonate, whereby $CaCO_3$ is precipitated. From the filtered clear solution caustic soda throws down a copious precipitate of $CaCO_3$.

The reaction appears to be so delicate that it may be utilized in determining the percentage of Na_2CO_3 in $NaHCO_3$.

If a hyposulphite solution contains both lead and calcium, it becomes possible to separate lead carbonate by sodium bi-carbonate, retaining calcium in solution. Calcium carbonate would only be precipitated in case the solution was caustic. The subject needs further theoretical and practical investigation.

§ 1805. *Results at the Yedras Mill, Sinaloa, Mexico.*

At the end of § 1805, I state that the small loss in sodium hyposulphite at the Yedras Mill is abnormal and a chemical paradox. I have now found a very simple and satisfactory explanation of this apparently obscure problem. The whole question turns on the conversion of sodium tetrathionate into hyposulphite during any stage of the lixiviation-process.

I find in Graham-Otto's Chemistry the following reaction:

$$Na_2S_4O_6 + Na_2S = 2Na_2S_2O_3 + S.$$

I consulted several eminent chemists, but not one of them seemed to be aware of the existence of this reaction. It had previously escaped my notice in reading the chapter on thion acids. Hence, the statement on page 136: "*It is absolutely false to assume that sodium hyposulphite converted into tetrathionate is again regenerated in the precipitating-tanks,*" must be retracted and modified.

That the conversion of $Na_2S_4O_6$ into $Na_2S_2O_3$, by Na_2S or Na_2S_2, in the precipitating-tanks can not be a very complete one is hardly necessary to point out. In adding sodium sulphide, the metals are, of course, precipitated first on account of their greater affinity for sulphur. After the solution is nearly free from silver, lead and copper, the conditions for the conversion of sodium tetrathionate into hyposulphite become more favorable; but at this point of the operation the reagent must be added sparingly and not in excess.

A calculation shows that the production of one pound $Na_2S_2O_3 + 5aq.$ from $Na_2S_4O_6$, by Na_2S or Na_2S_2, requires only 0.278 pounds commercial caustic soda containing 87 per cent. NaHO. This refers to the quantity of caustic soda consumed in the manufacture of sodium sulphide.

This subject is very interesting and important, and needs further investigation.

While this theory fully explains the small loss in sodium hyposulphite at Yedras, it remains to account for the much greater loss of this salt, as recorded in § 1202, page 136, in lixiviating "*acid*" and "*alkaline*" ores.

The explanation is contained in § 307 to § 315. There I have clearly shown the conditions producing sulphates and tetrathionates in the decomposition of extra-solutions. In the application of standard extra-solution to Yedras ore, the liquor is allowed to stand in the charge, while in treating acid ores it is circulated. In the former case the decomposition of the extra-solution will be very slight; in the latter case it must be considerable. In circulating the solution the conditions for the formation of both sodium tetrathionate and sulphate are very favorable. The liquor is brought repeatedly in contact with the atmosphere, thus producing tetrathionate; and the hot steam of the Korting ejector causes decomposition of a part of the extra-solution, forming sulphate.

In comparing acid ores with alkaline ores, the loss in sodium hyposulphite in the latter case is smaller for the same reason. That more sodium hyposulphite is lost in treating alkaline ores than arsenical alkaline ores, (Yedras) can also be explained. The extra-solution applied to alkaline ores is not of standard composition, but contains an excess of sodium hyposulphite. Solutions of such composition seem to have a greater tendency to form sulphates by atmospheric decomposition, at ordinary temperature, than standard extra-solution. Besides, the volume of the solution is much greater, and it is kept in motion, although not circulated.

Finally, it becomes evident that the proposition discussed in § 322, namely, the preparation of extra-solution by dissolving the $\frac{2}{3}$ salt of Lenz in sodium hyposulphite, falls to the ground.

Although this discussion does not by any means exhaust the subject, it must be considered an important addition to the chemistry of the Russell process.

CARL A. STETEFELDT.

NEW YORK, Sept. 6th, 1888.

ANALYTICAL LABORATORY

AND

ASSAY OFFICE

OF

Dr. GIDEON E. MOORE

Analytical and Consulting Chemist,

69 LIBERTY ST. - - - - - - - - NEW YORK.

Assays and Analyses of Ores, Minerals, Metals, Fertilizers, Waters, Products of the Arts, etc., etc.

Investigations made and advice furnished in the development of new metallurgical and manufacturing processes and in the prosecution of all departments of chemical and metallurgical industries, and manufactures involving the applications of chemistry.

Expert evidence in chemical questions before the courts.

Special arrangements made with manufacturing establishments, metallurgical works, etc., to furnish advice and analytical work by the year.

Consultations by letter and samples sent by mail or express will receive prompt attention.

Charges on samples sent by freight or express must be prepaid.

☞ Printed price-lists on application. ☜

FRASER & CHALMERS,

Fulton and Union Streets, Chicago, Ill.

MINING AND MILLING MACHINERY

OF EVERY DESCRIPTION.

Complete Plants furnished for the systematic treatment of all ores, on latest improved designs, by

LEACHING, AMALGAMATION, CONCENTRATION AND SMELTING.

Improved fast-running ROLLS — all known successful ROASTING FURNACES and dry kilns—CRUSHERS, STAMPS, PANS, JIGS, BUDDLES, FRUE VANNERS and EMBREY CONCENTRATORS.

Huntington Roller Mills, Sturtevant Mills, Water Jacket Furnaces.

Boss continuous process of Amalgamation — Hallidie Wire Rope Tramways.

Corliss Engines and Boilers.

Stetefeldt Roasting Furnaces and Dry Kilns.

REFERENCE TO FOLLOWING PLANTS BUILT:

Anaconda Copper Co.,	Mon.,	10	steam	stamps.	Segui Mill,	Uruguay,	20	stamps,	gold.
Granite Mountain Co.,	"	160	stamps,	silver.	Gregorio Mini'g Co.,	Uruguay,	40	"	"
Bi-metallic Mining Co.,	"	40	"	"	Guanandi Mining Co.,	"	20	"	"
Montana Mining Co.,	"	110	"	"	Uruguay Gold Mg. Co.,	"	30	"	"
Blue Bird Mining Co.,	"	60	"	"	Société St. Elie Fr'nch G'yana,		20	"	"
Empire Mining Co.,	"	60	"	gold.	Cie La Mana	" "	20	"	"
Homestake Cos., Dakota,		550	"	"	El Callao, Venezuela,		60	"	"
Rio del Oro (A), Arg. Rep.,		25	"	"	V'nezuel'n Austin, Venezuela,		40	"	"
" mill (B),	"	25	"	"	El Choco,	"	20	"	"
" " (C),	"	25	"	"	La Union,	"	20	"	"
" " (D),	"	25	"	"	Backus & Johnston, Peru, Concentrator.				
Co Todos Santos, Chili,		10	"	silver.	Paarl Pretoria, Africa, Gold Mill.				
Broken Hill Prop., Australia,		3	steam	stamps.					

BRANCH OFFICES:

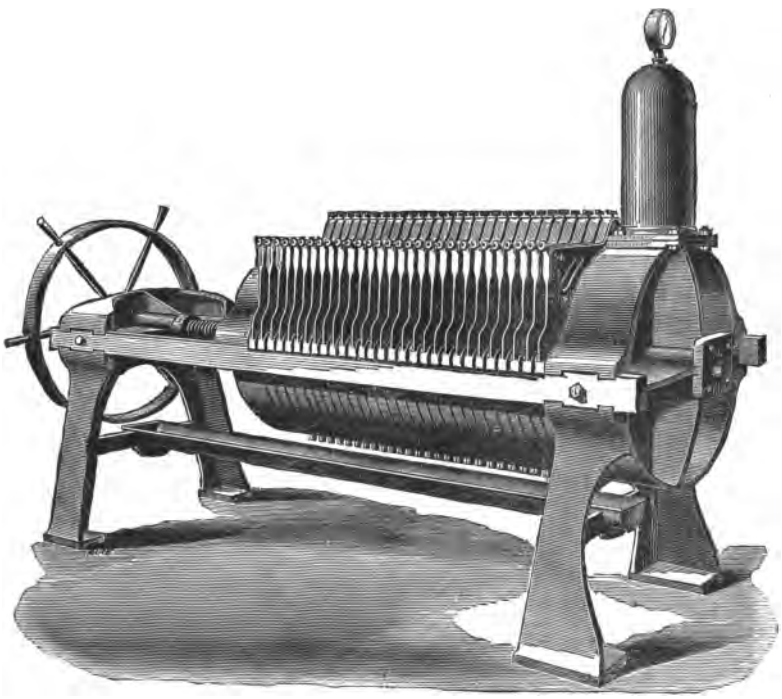
NEW YORK, 2 Wall street. LONDON, 23 Bucklersbury, E. C.

DENVER, 1316 18th street. SALT LAKE CITY, 7 W. 2d S. st.

MEXICO, 11 calle de Juarez, Chihuahua City.

Also at JOHANNESBERG, South Africa, and LIMA, Peru.

THE "JOHNSON" FILTER-PRESS.



JOHN JOHNSON & CO.,

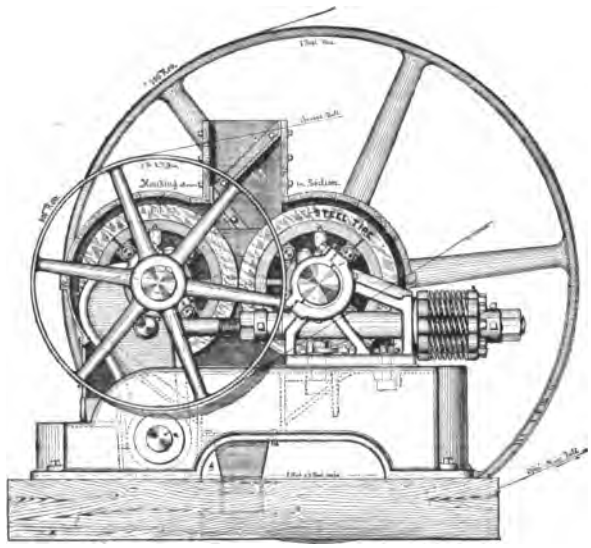
FRANKLIN SQUARE,

NEW YORK.

MAKERS AND PATENTEES OF FILTER-PRESSES

For perfect Filtration of Liquids; separating, collecting and pressing the Solids therefrom. Also arranged for Lixiviating or Leaching soluble matter from the solids collected, while in the Filter-press, in one operation. Also makers of Forcing Receivers for Filling Filter-presses without the use of pump.

☞ SEND FOR CATALOGUE. ☜



KROM'S PERFECTED STANDARD

STEEL ROLLS & ORE-BREAKERS.

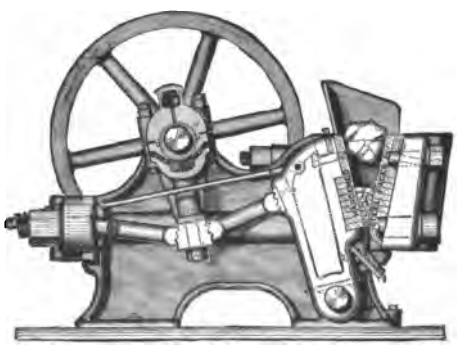
Revolving Screens, Ore-Feeders,
Dry-Kilns and Ore Concentrators.

Laboratory Rolls, Ore-Breakers,
Ore Concentrators and Screens.

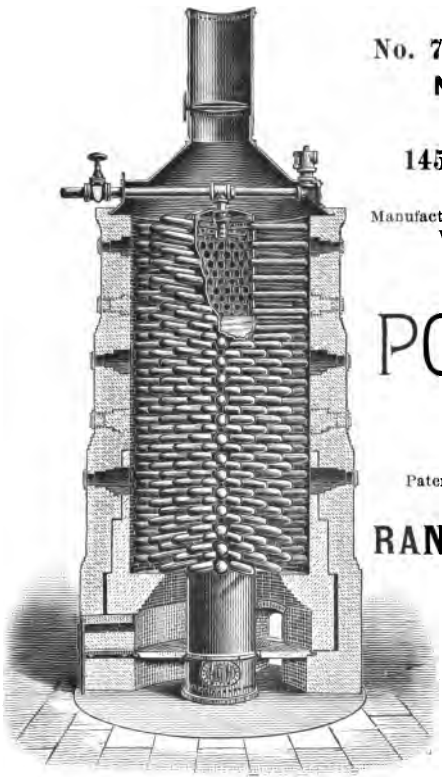
Plans for Lixiviation, Concentrating
and Ore-Crushing Works.

S. R. KROM,

151 Cedar Street, New York, U. S. A.



THE HAZELTON BOILER CO.,



Standard 215 Horse Power Stationary Boiler, with one-half of Brickwork, Smoke-Hood and Smoke-Stack removed.

The Largest Volume of Very Dry Steam, with Least Consumption of Fuel.

Admirably Adapted to the Use of Any Refuse Combustible Matter.

ABSOLUTELY SAFE
Under Unusually High Pressures.

LIGHT IN WEIGHT.
VERY DURABLE.

No Brick Chimneys Required.
Accessible in Every Part for Interior and Exterior Examination and Cleaning. Easily Repaired.

General Office:
No. 716 EAST 13th STREET,
NEW YORK, U. S. A.

Branch Office:
145 Broadway, New York,
Room 28.

Manufacturers and Sole Proprietors of that type of Water-Tube Boiler frequently called

THE PORCUPINE BOILER,

Patented in the United States and Foreign Countries.

RANKIN, BRAYTON & CO.

Proprietors of Pacific Iron Works,
SAN FRANCISCO, CAL.

MANUFACTURERS OF GENERAL MINING MACHINERY AND SOLE MANUFACTURERS OF THIS BOILER FOR CALIFORNIA, NEVADA, IDAHO, OREGON, WASHINGTON, ALASKA, ARIZONA, LOWER CALIFORNIA, BRITISH COLUMBIA AND WEST COAST OF MEXICO.

THE HENDEY AND MEYER ENGINEERING CO.,

DENVER, COL.

Manufacturers of General Mining Machinery and Sole Manufacturers of this Boiler for Colorado, Montana, Wyoming, Utah, New Mexico and East Coast of Mexico.

We are prepared to erect Plants of Boilers of any desired capacity, and solicit inspection of those now in operation. Catalogues giving full information sent free on application.

CORRESPONDENCE SOLICITED.

MINING PUMPS A SPECIALTY.

Send for special catalogue of Knowles' Mining Pumps, etc.

PUMPING MACHINERY FOR WATER WORKS

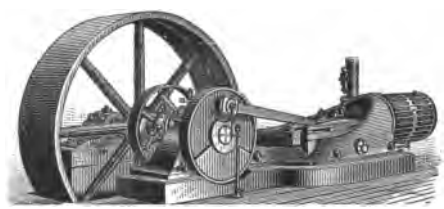
and every other service.

Knowles' Air Compressors for elevating acid solutions by air pressure.

93 Liberty Street, - - - - - New York.

BUCKEYE ENGINE. CO.,

Salem, Ohio.



W. L. SIMPSON,

General Eastern Sales Agent,

301 TELEPHONE BUILDING, 18 CORTLANDT ST., NEW YORK.

ENGINES OF ANY SPEED from 75 to 250 revolutions.

ENGINES OF ANY POWER from 13 to 1000 H. P.

SULPHUR.

CRUDE SULPHUR, FLOWERS OF SULPHUR,
GROUND SULPHUR, VIRGIN ROCK,

FOR

ACID WORKS, SHEEP DIP, SULPHURIZING VINES

AND FOR

Preparation of Sodium and Calcium Sulphide

USED IN THE

LIXIVIATION-PROCESS.

DICKERT & MYERS SULPHUR CO.,
P. O. Box 768, - - - Salt Lake City, Utah.

THE SOLVAY PROCESS CO.,

Syracuse, N. Y.,

MANUFACTURERS OF

SODA-ASH, BI-CARBONATE

AND

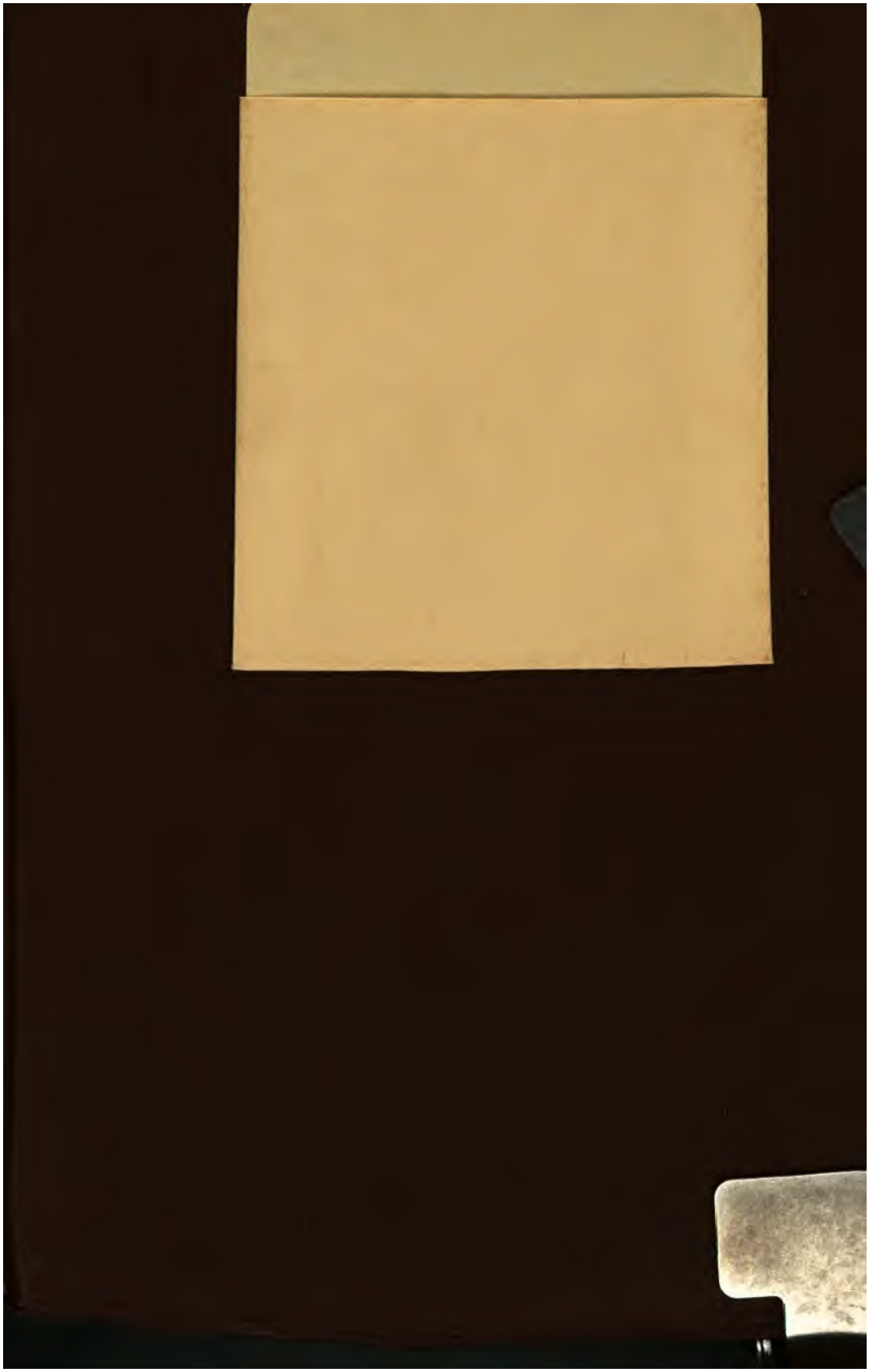
CAUSTIC SODA.

We guarantee test and quality of every shipment. The best and most uniform Soda in the market.

WING & EVANS, SOLE AGENTS,
54 William Street, - - - New York City.

ENGINEERING
LIBRARY

b89083894808a



B89083894808
B89083894808A